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CANADA
DEPARTMENT OF MINES

HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH

JOHN MCLEISH, DIRECTOR

INVESTIGATIONS IN ORE DRESSING
AND METALLURGY

(Testing and Research Laboratories)

1923 - 1925

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Sulphides

Native Gold and Gold Telluride

Polished specimen, Wright Hargraves No. 1 ore natural size. Showing native gold associated with gold telluride. Chalcopyrite and pyrite are also present. The sides marked A and B are slickensided and show amorphous molybdenite. The serious fracturing of the ore is also illustrated.

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OTTAWA
F. A. ACLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
1925

No. 617

Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

Investigations of Mineral Resources and the Mining Industry.

Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).

Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).

Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.

MINES BRANCH INVESTIGATIONS IN

ORE DRESSING AND METALLURGY, 1923

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

During the year 1923, the laboratories of the Division were devoted entirely to experimental and research work in connexion with the treatment of Canadian ores and metallurgical products. The results of the various investigations are given in the reports by the officials of the Division directly engaged on the work. A brief review of the investigations is given below.

Report No. 185

THE CONCENTRATION OF THE ORE FROM THE ROSSLAND VELVET MINES, LTD.,
ROSSLAND, B.C., BY R. K. CARNOCHAN

The experimental work showed that the ore could be concentrated with good recoveries of the gold and copper values by table concentration and flotation, or by flotation alone. Tabling and flotation gave the higher recoveries; straight flotation gave the higher grade product.

Report No. 186

THE CONCENTRATION OF THE ORE FROM THE COLUMBIA-KOOTENAY MINE,
ROSSLAND, B.C., BY R. K. CARNOCHAN

The shipment submitted was very low grade, badly weathered and oxidized. The experimental tests, however, showed that the copper and gold values could be concentrated by table concentration and flotation, or by straight flotation. On higher grade ore, results similar to those obtained on other Rossland ores are possible.

Report No. 187

SELECTIVE FLOTATION TESTS ON SULLIVAN MINE ORE, KIMBERLEY, B.C.,
BY C. S. PARSONS

The experimental work on Sullivan ore was done in order to obtain a comparison of results using various flotation reagents. Tests were made using reagents other than those used at the Trail plant and using their reagents, for a comparison of results. From the results it was found that Z cake and Fumol were very satisfactory for the flotation of the lead, and light hardwood creosote and No. 34 gravity fuel oil gave good results for the flotation of the zinc. Better results were obtained with these reagents than by using the reagents in practice at Trail.

Report No. 188

FLOTATION TESTS ON COPPER MOUNTAIN ORE, COPPER MOUNTAIN, B.C.,
BY R. K. CARNOCHAN

The test work was done to determine the flotation reagents suitable for use in the concentration of this ore by flotation. The copper minerals present being chalcopyrite and bornite, permits the production of a high-grade concentrate. The recovery of the copper values varied with the degree of fineness to which the ore was ground. Of the flotation reagents used, TT mixture gave the higher grade concentrates, showing a selective action for the copper minerals.

Report No. 190

CONCENTRATION OF A COPPER GOLD ORE FROM SURF INLET, B.C.,
BY R. K. CARNOCHAN

The test work on the Belmont Surf Inlet ore was done in order to determine the results obtained by different methods of treatment, and to study the effects of different oil mixtures used in the flotation of the ore. Table concentration and flotation gave much better results than flotation alone. High recoveries of the gold, silver, and copper values were made in a high-grade concentrate. Of the flotation reagents used, TT mixture had a better selective action for the copper minerals, giving a higher grade concentrate than the other flotation reagents.

Report Nos. 191-192

GOLD ORE FROM CRANBERRY HEAD, YARMOUTH CO., N.S., BY C. S. PARSONS

Experimental test and research work was done on two shipments of ore, to obtain the necessary data for determining the best metallurgical treatment. It was found that amalgamation was essential in the treatment of this class of ore; that the amalgamation tailing could be cyanided without difficulty, with the recovery of practically all the remaining gold values, or that the amalgamation tailing could be concentrated on tables with the production of a concentrate, marketable for its gold and arsenic content, or the gold recovered by pan amalgamation and the residue marketed for its arsenic and gold content. The reports describe in detail the various methods that could be applied, and give the results obtained from each method. The investigator has described very fully the different methods that could be applied to Nova Scotia gold ores.

Report No. 193

AMALGAMATION AND CYANIDE TESTS ON THE ORE FROM THE OPHIR LODGE
MINE, B.C., BY R. K. CARNOCHAN

Experimental tests were made to determine if amalgamation at 80 mesh was preferable to amalgamation at 40 mesh, and if it would be necessary to cyanide after amalgamation. It was found that amalgamation at 40 mesh was as efficient as amalgamation at a finer mesh and that cyanidation was necessary after amalgamation to obtain a high recovery of the gold values. The results show high recoveries by amalgamation and cyanidation.

*Report No. 194*THE CONCENTRATION OF THE COMPLEX SULPHIDE ORE OF THE EUSTIS MINE,
QUEBEC, BY C. S. PARSONS

The experimental work was done on a carload shipment, and consisted of making semi-commercial runs to determine the correct conditions for the selective flotation of the chalcopyrite from the pyrite, in this heavy sulphide ore. The results showed that a copper concentrate containing better than 20 per cent copper could be made with a recovery of over 90 per cent of the copper values. These results were obtained by accurate control of the proper conditions in the ball mill, classifier and flotation circuits, of the grinding, pulp density, reagents, etc.

*Report No. 195*EXPERIMENTAL TESTS ON GOLD ORE FROM THE KIRKLAND LAKE GOLD MINING
CO., KIRKLAND LAKE, ONT., BY R. K. CARNOCHAN

Tests were made on three samples, one of high-grade ore containing molybdenite and tellurides, one of very low-grade ore, and one of mill tailings. The experimental work on the high-grade sample was conducted to determine whether the presence of molybdenite and tellurides had any effect on the recovery of the gold values, and what recoveries could be obtained. Considering the grade of the ore, good recoveries were made by cyanidation. The molybdenite does not seem to have any deleterious effect on the recovery of the gold by the cyanide process. The presence of tellurides requires that the ore be ground extremely fine for cyanidation. Cyanidation gave good results on the low-grade ore, and also on the tailing sample. The high tailings seem to be due to the coarseness of the grinding, as shown in the screen tests.

*Report No. 196*EXPERIMENTAL TESTS ON GOLD ORE FROM THE WRIGHT HARGREAVES MINES,
LTD., KIRKLAND LAKE, ONT., BY R. K. CARNOCHAN

Tests were made on a high-grade sample, containing molybdenite, tellurides and free gold, to determine whether the presence of molybdenite and tellurides had any effect on the recovery of the gold values. Considering the grade of the sample, good recoveries were made by cyanidation. The molybdenite does not seem to have any deleterious effect on the recovery of the values. The presence of tellurides requires that the ore be ground extremely fine for good extraction of the values by cyanidation.

Included in the report is a microscopic examination of selected specimens of the ore by E. A. Thompson, showing the presence of the various minerals.

*Report No. 197*GOLD ORE FROM DASSERAT ROUYN GOLDFIELDS, LTD., ROUYN TOWNSHIP,
TÉMISCAMINGUE DIST., QUE., BY R. K. CARNOCHAN

Experimental tests were made to determine whether the ore was amenable to treatment with high recoveries of the precious metal values. The tests show the ore as submitted to be very amenable to treatment by the cyanide process. A very high extraction of the values was obtained.

Report No. 198

CONCENTRATION TESTS ON GRAPHITE FROM THE CANADIAN GRAPHITE CORP.,
GUENETTE, QUE., BY C. S. PARSONS AND R. K. CARNOCHAN

Tests were made to check up the results being obtained in the company's concentrator and to determine the percentages of recoverable graphite in as coarse a form as possible, with high recoveries of the graphite. The tests show that there should be no difficulty in recovering a good percentage of the graphite in coarse flake; that good recoveries can be made; that high-grade flake can be produced by flotation and subsequent refining of the flotation concentrate. The point to which concentration should be carried before refining operations begin, is discussed in the report.

Report No. 199

SELECTIVE FLOTATION OF THE COPPER-NICKEL ORE OF SHEBANDOWAN LAKE
DISTRICT, ONTARIO, BY C. S. PARSONS

Experimental tests were conducted to determine whether the ore could be concentrated and a high-grade concentrate obtained carrying the copper-nickel, cobalt, and precious metal values, with good recoveries of these values. The tests show that the ore can be concentrated by selective flotation, with the production of a concentrate assaying 15 per cent in copper, nickel, and cobalt, with a recovery of 90 per cent of these values. Further work will be conducted to determine whether the precious metal values are reporting in the flotation concentrate.

Report No. 200

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE PROPERTIES OF THE
MOLYBDENITE REDUCTION CO., NEAR AMOS, QUE., BY C. S. PARSONS

The object in view in conducting experimental tests was to determine whether the ore was amenable to treatment by the flotation of the molybdenite from the gangue minerals, with the production of a high-grade marketable product with a high recovery of the molybdenite values. The results showed the ore to contain an average of 2 per cent molybdenite from which a high-grade concentrate, averaging 90.8 per cent molybdenite was obtained with a recovery of 95 per cent of the molybdenite values in the ore. The tailings contained 0.10 per cent molybdenite. No difficulty was experienced in obtaining the above results which were from tonnage check tests.

Report No. 201

THE CONCENTRATION AND TREATMENT OF ARSENICAL GOLD ORE FROM
BATHURST, N.B., BY J. S. GODARD

Gravity concentration tests were carried out by means of jigs and tables, and by flotation, to determine whether a high-grade arsenical concentrate could be produced from the ore. Flotation gave the better results, producing a concentrate containing 30 per cent arsenic with a recovery of 90 per cent of the arsenic values. The ore contains small amounts of gold and silver values, about \$2.50 per ton. The arsenopyrite concentrates contain \$3 to \$3.50 per ton in gold and silver. Leaching tests made on the concentrates showed that the cost of leaching and consumption of cyanide was too high to make the extraction of precious metal values by this method feasible.

Report No. 202

THE TAILING DUMPS OF THE NAUGHTON GOLD MINE, NAUGHTON, ONT., BY
C. S. PARSONS

The report contains the estimated reclaimable tonnage in the dumps, the average assay of arsenic and gold content, and the results of simple concentration tests. The assays and results of the concentration tests indicate that it will not pay, at present, to reclaim and concentrate the dumps, especially the lower dump, as the grade and amount of concentrate obtainable would be too low to stand the cost of concentration, treatment charges, and transportation to the smelter.

Report No. 203

CONCENTRATION OF THE ZINC-IRON MIDDLING FROM THE DUMP AT NOTRE-DAME DES ANGES, QUE., BY C. S. PARSONS

The purpose of the experimental work was to determine whether the zinc, lead, gold and silver values could be concentrated by the elimination of the iron sulphides and any gangue minerals. Magnetic separation and flotation tests were made. By magnetic separation after giving the material a magnetic roast, a high recovery of the zinc was obtained. The zinc concentrate contains all the siliceous material and this prevents a high-grade zinc product from being made. Table concentration would improve the grade of the zinc product. The flotation tests indicate that a 40 per cent zinc product can be obtained by selective flotation of the zinc from the iron sulphides, but attention is drawn to the difficulty in obtaining and maintaining the correct flotation conditions in the treatment of a limited tonnage of partly oxidized dump material.

Section III

The investigations carried on by the staff of the chemical laboratories of the Division, were as follows:—

THE VALUE OF GIVING THE SILVER RESIDUES FROM THE DOMINION REDUCTION COMPANY, COBALT, ONT., A WEAK SULPHURIC ACID WASH PREVIOUS TO CYANIDING, BY B. P. COYNE.

It was found that a 1 per cent sulphuric acid wash was as effective as a stronger acid wash for removing soluble salts; that it was easier to wash out the soluble salts when a weak solution was used, and that this treatment reduces the cyanide consumption.

DETERMINATION OF SOLUBLE CYANIDES IN CRANBERRY HEAD GOLD ORE, AND THE EFFECT OF CERTAIN REAGENTS ON CYANIDE CONSUMPTION, BY B. P. COYNE.

It was found that the ore does not contain any cyanides soluble under working conditions, and that the use of Na_2O_2 reduced the cyanide consumption.

DETERMINATION OF SOLUBLE SALTS IN EUSTIS COPPER ORE, AND THE EFFECT OF ADDING LIME FOR SELECTIVE FLOTATION OF THE CHALCOPYRITE, BY B. P. COYNE.

It was found that the oxidized ore contained a much greater amount of soluble salts than the unoxidized ore, and that the addition of lime prevented the iron from going into solution.

THE PRECIOUS METAL VALUES IN PRODUCTS FROM SMELTING TESTS ON COPPER-NICKEL ORE OF SHEBANDOWAN LAKE, ONT., BY H. C. MABEE.

The matte produced was low grade, representing 37.5 per cent of the weight of the ore used. Analysis of the matte showed it to contain gold 0.03 oz., platinum 0.085 oz., palladium 0.131 oz., and that almost all the total precious metal values in the ore were contained in the matte.

THE ELIMINATION, BY SELECTIVE FLOTATION, OF BARREN PYRRHOTITE FROM THE NICKELIFEROUS PYRRHOTITE ORES OF THE SUDBURY DISTRICT, BY H. C. MABEE.

A study was made of the flotation products to determine the amounts of barren pyrrhotite that were eliminated in the tailings by selective flotation of the ores. This was found to be 30 to 50 per cent of the pyrrhotite in the ores. This elimination of barren pyrrhotite, together with the gangue minerals, is significant, and means a saving in several directions if the ores were concentrated before smelting.

THE HYDROMETALLURGICAL TREATMENT OF PYRRHOTITE AND PYRITE ORES WITH THE PRODUCTION OF ELECTROLYTIC IRON, AND THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS, BY R. J. TRAILL.

The results of the test work carried on during the year are given in the report. The test work was carried to a point where better facilities for carrying on the investigation were required. A new laboratory was built and is being equipped, so that a complete cycle of operations can be maintained. Although the equipment is on a small scale and only small scale tests can be conducted, it was deemed advisable to work out the problems on a small laboratory scale before embarking on any extensive large scale work. The process has possibilities of being commercially applicable to Canadian pyrrhotite and pyrite ores.

Section IV

As a large part of the experimental work done in the laboratories is on the flotation of ores, a summary of the work on selective flotation as applied to Canadian ores was compiled by C. S. Parsons. This report is inserted as Section IV.

Section V

A considerable amount of experimental work has been done in the laboratories on the beneficiation of Canadian iron ores. A summary of the more important tests was compiled by W. B. Timm. It is inserted under Section V.

Section VI

During the last two years, experimental tests were made on the selective flotation of the lower grade nickeliferous pyrrhotite ores of Ontario. The ores were from the Sudbury nickel area and the Shebandowan Lake district of Ontario. The results of the various tests have been collected and compiled in one report. It is inserted as Section VI.

Section VII

During the year a number of ore concentration plants in Canada were visited by officials of the Division, and a study made of the processes being applied to the concentration of Canadian ores. A description of some of the important ones is included in this section.

LIST OF ORES AND METALLURGICAL PRODUCTS ON WHICH
EXPERIMENTAL TEST AND RESEARCH WORK
WAS CONDUCTED

In the following table is given a list of the ores and metallurgical products received, with the report number, class of ore or product, source of shipment, shipper and weight of shipment, on which experimental test and research work was conducted:—

Report No.	Ore or product	Source of shipment	Shipper and address	Weight pounds
185	Gold-copper..	Rossland, B.C.....	Rossland Velvet Mines, Ltd., Rossland, B.C.....	160
186	Gold-copper..	Rossland, B.C.....	Columbia-Kootenay Mines, Ltd., Rossland, B.C.....	60
187	Lead-zinc.....	Kimberley, B.C.....	Cons. Mining and Smelting Co. of Canada, Ltd., Kimberley, B.C.....	100
188	Copper.....	Copper Mountain, B.C..	Canada Copper Corporation, Copper Mountain, B.C.....	100
190	Gold-copper..	Surf Inlet, B.C.....	Belmont Surf Inlet Mines, Ltd., Surf Inlet, B.C.....	150
191	Gold.....	Cranberry head, Yarmouth co., N.S.....	S. F. Johnson, Boston, Mass., U.S.A....	280
192	Gold.....	Cranberry head, Yarmouth co., N.S.....	S. F. Johnson, Boston, Mass., U.S.A....	380
193	Gold.....	Ophir Lode mine, Lardau, B.C.....	H. B. Morrison, Nelson, B.C.....	20
194	Copper.....	Eustis, Que.....	Eustis Mining Co., Eustis, Que.....	40,000
195	Gold.....	Kirkland Lake, Ont....	Kirkland Lake Gold Mining Co., Kirkland Lake, Ont.....	200
196	Gold.....	Kirkland Lake, Ont....	Wright Hargreaves Mines, Ltd., Kirkland Lake, Ont.....	150
197	Gold.....	Rouyn township, Témiscamingue dist., Que.	Dasserat-Rouyn Gold Mines, Ltd., Ottawa, Ont.....	950
198	Graphite.....	Guenette, Que.....	Canadian Graphite Corporation, Guenette, Que.....	300
199	Copper-nickel	Shebandowan lake, Ont.	Jamieson and Peacock, Duluth, Minn., U.S.A.....	400
200	Molybdenite..	La Corne and Malartic tps., Abitibi dist., Que.	Molybdenite Reduction Co., Montreal, Que.....	20,186
201	Arsenic-gold..	Bathurst, N.B.....	L. D. Densmore, Bathurst, N.B.....	200
202	Arsenic-gold..	Naughton, Ont.....	Mines Branch, Department of Mines, Ottawa, Ont.....	200
203	Zinc.....	Notre-Dame des Anges, Que.....	British Metals Corporation, Montreal, Que.....	110

II

REPORTS ON INVESTIGATIONS CONDUCTED

Report No. 185

THE CONCENTRATION OF THE COPPER-GOLD VALUES IN THE ORE
FROM THE ROSSLAND VELVET MINES, ROSSLAND, B.C.

R. K. Carnochan

Shipment.—A shipment of ore, 157½ pounds net, was received at the Ore Dressing and Metallurgical Laboratories, October 12, 1922, from the Rossland Velvet Mines, Ltd., Rossland, B.C.

Purpose of experimental tests.—The ore contains values in gold, silver, and copper, and tests were desired to determine a suitable method of concentrating these values.

Analysis of shipment.—The shipment was divided into three lots. Samples of these lots were taken and found to contain:—

	Gold oz. per ton	Silver oz. per ton	Copper per cent
Milling ore.....	0.26	0.32	2.58
No. 1 dump ore.....	0.18	0.26	1.01
Oxidized dump ore.....	0.26	0.34	1.12

Experimental Tests

Test No. 1—No. 1 dump ore; tabling and flotation.—A lot of 2000 grammes of No. 1 dump ore —20 mesh, was screened on 80 mesh. This gave:—

—20+80.....	1,310 grammes
—80.....	690 “

The —20+80 was run over a small Wilfley table making a concentrate, middling, and tailing. The —80 and the table middling were ground for 15 minutes in a small ball mill with 25 drops of water-gas tar and then floated in a small Ruth machine with 2 drops of P.T.T. No. 350.

Test No. 2—Milling ore; tabling and flotation.—A lot of 1417 grammes of milling ore, —20 mesh, was screened on 40 and 80 mesh. This gave:—

—20+40.....	640 grammes
—40+80.....	256 “
—80.....	491 “

The three sizes were tabled separately making in each case a concentrate and a tailing. The —20+40 table tailing was ground for 5 minutes in a small ball mill with 25 drops of water-gas tar, the —40+80 tailing was then added and the ball mill run for 5 minutes more, the —80 tailing was now added and the charge ground again for 5 minutes. The ball mill charge was then floated in a small Ruth machine with 5 drops P.T.T. No. 350.

Test No. 3 on oxidized dump ore; tabling and flotation.—A lot of 1655 grammes of oxidized dump ore -40 mesh was screened on 80 mesh. This gave:—

-40+80.....	690 grammes
-80.....	965 "

Each of the above sizes was tabled, making a concentrate and a tailing. The -40+80 table tailing was ground for 5 minutes in a small ball mill with 25 drops of water-gas tar. The -80 tailing was then added to the mill and the whole charge ground for 5 minutes more. The ball mill charge was then floated in a small Ruth machine with 10 drops of P.T.T. No. 350.

Test No. 4 on No. 1 dump ore; tabling and flotation.—This test is similar to Test No. 2.

-20 +40.....	692 grammes
-40 +80.....	379 "
-80.....	557 "

The flotation concentrate was re-run to clean it up. This gives from flotation a concentrate, a middling, and a tailing.

Test No. 5 on milling ore; flotation.—The ore, -20 mesh, was ground for 30 minutes with 25 drops of water-gas tar in a small ball mill and then floated in a small Ruth machine with 3 drops P.T.T. No. 350. The concentrate was re-run to clean it up. This gives 3 products from flotation.

Test No. 6 on No. 1 dump ore; flotation.—Similar to test No. 5.

Test No. 7 on oxidized dump ore; flotation.—Similar to test No. 5.

Test No. 8 on oxidized dump ore; flotation.—The ore, -20 mesh, was ground for 30 minutes in a small ball mill with 0.6 c.c. heavy hardwood creosote oil (F.P.L. 26) and 0.4 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ball mill charge was then floated in a small Ruth machine, the concentrates being re-run to clean them.

Test No. 9 on oxidized dump ore; flotation.—The ore, -20 mesh, was ground for 30 minutes in a small ball mill with 3 grammes soda ash and then floated in a small Ruth machine with 10 drops of TT (Callow alphabetical reagent). The concentrate was re-run to clean it.

Test No. 10 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that 3 grammes of lime were used in place of the soda ash and 10 drops of XY were used in place of the TT.

Test No. 11 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that lime was used in place of soda ash, and thiofizzan in place of TT.

Test No. 12 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that lime was used in place of soda ash.

Test No. 13 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that XY was used in place of TT.

Test No. 14 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that thiofizzan was used in place of TT.

Test No. 15 on milling ore; flotation.—This test is similar to test No. 9. Three grammes of lime were used in grinding the ore, and 5 drops of TT were used for flotation.

Test No. 16 on No. 1 dump ore; flotation.—This test is similar to test No. 9. Three grammes of lime were used in grinding the ore and 5 drops of TT used for flotation.

Results of Experimental Tests

Test No.	Product	Weight grms.	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
1	Table concentrate.....	169	1.02	0.86	3.92	64.9	51.1	73.6
	Flotation concentrate.....	67	1.24	1.38	12.17			
	Table tailing.....	684	0.06	0.08	0.33			
	Flotation tailing.....	1,070	0.09	0.16	0.28			
	Loss.....	10	0.09	0.16	0.28			
2	Table concentrate +40.....	83	0.96	0.94	7.30	84.5	79.5	91.0
	“ “ +80.....	57	0.98	0.74	8.65			
	“ “ -80.....	72	0.84	0.88	10.12			
	Flotation concentrate.....	69	0.74	1.18	16.79			
	“ tailing.....	1,015	0.04	0.06	0.26			
	Loss.....	121	0.04	0.06	0.26			
3	Table concentrate +80.....	173	0.66	0.38	3.37	74.7	55.2	91.0
	“ “ -80.....	165	0.80	0.46	3.20			
	Flotation concentrate.....	65	0.76	1.14	8.15			
	“ tailing.....	1,070	0.08	0.14	0.13			
	Loss.....	182	0.08	0.14	0.13			
4	Table concentrate +40.....	127	0.68	0.48	2.87	78.8	68.9	88.1
	“ “ +80.....	62	1.08	0.70	3.67			
	“ “ -80.....	70	0.78	0.56	3.25			
	Flotation concentrate.....	62	0.73	0.95	7.62			
	“ middling.....	145	0.47	0.63	0.40			
	“ tailing.....	1,027	tr.	tr.	0.10			
5	Loss.....	135	tr.	tr.	0.10			
	Flotation concentrate.....	115	0.98	0.80	16.78	57.7	54.6	77.2
	“ middling.....	153	0.30	0.26	1.91	23.5	23.6	11.7
	“ tailing.....	734	0.05	0.05	0.38	18.8	21.8	11.1
	Loss.....	734	0.05	0.05	0.38	18.8	21.8	11.1
6	Flotation concentrate.....	76	1.32	0.88	10.23	61.0	52.4	68.7
	“ middling.....	173	0.24	0.22	0.86	25.2	29.8	13.2
	“ tailing.....	758	0.03	0.03	0.27	13.8	17.8	18.1
7	Flotation concentrate.....	44	1.10	1.14	11.99	17.4	13.9	46.8
	“ middling.....	178	0.60	0.62	2.64	38.4	30.7	41.6
	“ tailing.....	768	0.16	0.26	0.17	44.2	55.4	11.6
8	Flotation concentrate.....	150	5.95	79.2
	“ middling.....	188	0.69	11.5
	“ tailing.....	655	0.16	9.3
9	Flotation concentrate.....	77	9.38	66.9
	“ middling.....	134	1.32	16.4
	“ tailing.....	781	0.23	16.7
10	Flotation concentrate.....	135	6.80	80.4
	“ middling.....	118	0.63	6.5
	“ tailing.....	744	0.20	13.1
11	Flotation concentrate.....	165	5.57	82.6
	“ middling.....	146.5	0.76	10.0
	“ tailing.....	695	0.12	7.4
12	Flotation concentrate.....	126	0.73	0.93	7.75	37.2	38.7	79.2
	“ middling.....	131	0.33	0.45	0.88	17.5	19.4	9.3
	“ tailing.....	746.5	0.15	0.17	0.19	45.3	41.9	11.5
13	Flotation concentrate.....	38	14.31	46.4
	“ middling.....	175.5	2.78	41.7
	“ tailing.....	775	0.18	11.9

Results of Experimental Tests—Concluded

Test No.	Product	Weight grms	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
14	Flotation concentrate.....	171.5	5.66	82.2
	“ middling.....	118	1.05	10.5
	“ tailing.....	713	0.12	7.3
15	Flotation concentrate.....	82.2	1.34	2.12	26.50	46.8	54.3	88.0
	“ middling.....	77.1	0.63	0.69	2.23	20.7	16.6	6.9
	“ tailing.....	849.7	0.09	0.11	0.15	32.5	29.1	5.1
16	Flotation concentrate.....	40.3	2.00	2.31	17.94	44.8	46.0	77.4
	“ middling.....	78.8	0.75	0.71	1.43	31.5	27.6	12.1
	“ tailing.....	889.3	0.05	0.06	0.11	23.7	26.4	10.5

Screen Test of Flotation Tailings

Test No.	+35	+48	+65	+100	+150	—150	+200	—200
	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent
2.....	0.4	2.4	15.8	27.0	9.7	44.7		
3.....			1.2	5.9	15.1	77.8		
4.....			10.0	25.6	13.6	50.8		
5.....			0.6	8.8	17.3		18.3	55.0
6.....			0.1	3.2	10.0		23.1	63.6
7.....			0.1	6.3	10.9		22.7	60.0

SUMMARY

Milling ore.—The results obtained on the milling ore are as follows:—

Test No.	Method used	Product	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
2	Tabling and flotation...	Total conc....	0.88	0.94	10.63	84.5	79.5	91.0
5	Oil flotation.....	Concentrate..	0.98	0.80	16.78	57.7	54.6	77.2
		Middling.....	0.30	0.26	1.91	23.5	23.6	11.7
15	Flotation with alpha-betical reagents.....	Concentrate..	1.34	2.12	26.50	46.8	54.3	88.0
		Middling.....	0.63	0.69	2.23	20.7	16.6	6.9

No. 1 Dump ore.—The results obtained on No. 1 dump ore are as follows:—

Test No.	Method used	Product	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
4	Tabling and flotation...	Total conc....	0.79	0.63	4.02	73.8	63.9	88.1
		Middling.....	0.47	0.63	0.40	21.2	31.1	4.0
6	Oil flotation.....	Concentrate..	1.32	0.88	10.23	61.0	52.4	63.7
		Middling.....	0.24	0.22	0.86	25.2	29.8	13.2
16	Flotation with alpha-betical reagents.....	Concentrate..	2.09	2.31	17.94	44.8	46.0	77.4
		Middling.....	0.75	0.71	1.43	31.5	27.6	12.1

Oxidized dump ore.—The results obtained on oxidized dump ore are:—

Test No.	Method used	Product	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
3	Tabling and flotation...	Total conc....	0.73	0.54	4.07	74.7	55.2	91.0
7	Flotation.....	Concentrate..	1.10	1.14	11.99	17.4	13.9	46.8
		Middling.....	0.60	0.62	2.64	38.4	30.7	41.6
8	Oil flotation.....	Concentrate..			5.95			79.2
		Middling.....			0.69			11.5
12	Flotation with alphabetical reagents.....	Concentrate..	0.73	0.93	7.75	37.2	38.7	79.2
		Middling.....	0.33	0.45	0.88	17.5	19.4	9.3

CONCLUSIONS

1. Tabling and flotation give higher recoveries and lower grade products than flotation alone.

2. Flotation with alphabetical reagents gives better products and a higher copper recovery than oil flotation, but the oil flotation gives better gold and silver recoveries.

3. Cyaniding at -200 mesh for 24 hours reduced the tailings from tests Nos. 2 and 3 to a trace in gold and silver.

4. The best method of treating this ore can only be decided by a careful consideration of freight rates and smelter charges.

Report No. 186

THE CONCENTRATION OF THE VALUES IN THE PYRRHOTITE FROM THE COLUMBIA-KOOTENAY MINE, ROSSLAND, B.C.

R. K. Carnochan

Shipment.—A shipment of 56 pounds of ore was received September 5, 1922, from the Columbia-Kootenay mine, Rossland, B.C.

Characteristics of shipment.—The chief mineral constituent was pyrrhotite, carrying small amounts of chalcopyrite, gold and silver. It was in a weathered and oxidized condition.

Purpose of experimental tests.—Tests were desired to determine if the copper and precious metal values could be concentrated to a marketable grade.

Analysis.—The ore was crushed to 4 mesh and a sample cut out. This sample gave upon analysis the following:—

Gold.....	0.06 oz. per ton	Copper.....	0.46 per cent
Silver.....	0.08 "	Iron.....	36.35 "
Sulphur.....	20.46 per cent	Insoluble.....	22.40 "

Experimental tests:

Tabling and flotation.—About 1,400 grammes of the ore, -20 mesh, was screened on 40 and 80 mesh. Each size was tabled separately on a small Wilfley table, making a concentrate and a tailing. The table tailings were mixed, ground in a ball mill, and then floated in a small Ruth machine. All products were dried, weighed, sampled and analysed. The table concentrates averaged 0.07 oz. gold per ton and 0.48 per cent copper. The flotation concentrate ran 0.12 oz. gold per ton and 2.65 per cent copper.

Flotation.—Nine flotation tests were made on the ore using different oils and reagents. In the best test, the concentrate ran 3.02 per cent copper and the copper recovery was 76.0 per cent. The next best test gave a concentrate of 1.80 per cent copper with a recovery of 84.5 per cent. The other tests gave very much poorer results.

Flotation and tabling.—Two tests were made to see if it would be possible to raise the grade of flotation concentrates by tabling. It was found that the table concentrate and tailing ran practically the same in copper.

CONCLUSIONS

1. The ore submitted contains very low values in copper, gold and silver, and even if it could be concentrated satisfactorily it could not be worked economically.

2. If ore of higher grade was found on the property it would likely be amenable to flotation, for with a higher head a higher concentrate and recovery would be obtained.

3. Tabling seems of little value in treating an ore of this kind, as due to the large sulphide content, 44 per cent of the heads go into the concentrate when tabled.

4. Cyaniding the tails from the tabling and flotation test at -200 mesh for 24 hours reduced them to a trace of gold and silver. With higher grade ore this method might be of use in treating flotation tailings, if they are too high in gold and silver to discard.

Report No. 187

FLOTATION TESTS ON SULLIVAN MINE ORE

C. S. Parsons

A shipment of 100 pounds of ore from the Sullivan mine, Kimberley, B.C., was received July 18, 1922, from the Consolidated Mining and Smelting Co. of Canada, Trail, B.C., for the purpose of comparing the results obtained from treating the ore with the reagents used in their practice, with those obtained from the use of other flotation reagents.

The accompanying table gives the results from a series of nine tests made by flotation. The first three of these tests were made on the ore as soon as it was received. No further work was done on the ore until December, when a test was run with the reagents now being used at the company's mill.

In the first three tests, a method of treatment was used that had previously been worked out for the separation of the lead and zinc in the North Star ore. Such excellent separation was obtained that it is doubtful whether it can be very much improved on by any method of straight flotation alone. No attempt was made to table the lead from the zinc concentrate, but no doubt this could be done and an increase in the recovery of the lead obtained. In running these tests it was necessary to make both a lead and zinc middling product, but in practice this would be returned continuously to the lead and zinc circuits, or treated separately for the recovery of the values.

Tests Nos. 1, 2 and 3.—The general procedure in making the tests was as follows:—

1,000 grammes of ore crushed dry to 30 mesh was placed in a small ball mill and in the case of the first test 0.1 gramme Z cake added together with $2\frac{1}{2}$ grammes lime and $\frac{1}{2}$ gramme of Selecto. The lime keeps down the iron and also tends to keep the zinc down, but the Selecto is added for that purpose. The charge was then ground to the required fineness and dumped into a Ruth flotation machine. A little Fumol was added to froth the cell and a lead concentrate was floated. The cell was then run out and the lead concentrate placed back in the cell and recleaned. A lead middling from this recleaning operation was obtained which was kept separate. The lead tailing, which in the meantime had settled, was decanted in order to get rid of the Selecto and lime.¹ Soda ash, about 5 pounds or less per ton and 1 pound per ton copper sulphate were added.² YZ mixture was then added and the dewatered pulp washed into the cell. The zinc was floated, a little Fumol being used to increase the froth. The zinc concentrate was recleaned, producing a zinc middling.

Considering the results obtained in the first three tests and from similar results obtained on the North Star ore, it appears that Z cake and Fumol are the best for floating the lead, and that light hardwood creosote oil from the Standard Chemical Co. and No. 34 gravity fuel oil from the Great North Western Oil Co., Cleveland, Ohio, are the most suitable oils for the zinc.

Test No. 6.—In this test the Selecto was not added, the object being to determine if a satisfactory separation of the lead and zinc could be obtained without it, and at the same time avoid the step of dewatering the lead tailing before attempting to float the zinc. The procedure was as follows:—

The ore was ground with 5 pounds per ton of lime and 1 pound per ton soda ash, and 0.1 gramme of Z cake. The cell was frothed to float the lead with Fumol No. 6. After eliminating the lead, the zinc was floated by adding 1 pound copper sulphate per ton, and K.K. oil and a little No. 5 pine oil to froth the cell. The results obtained were very poor. Referring to the table, it will be seen that 13.8 per cent of the zinc floated with the lead and that the lead concentrate was low grade. The zinc concentrate was very dirty and only a 10.4 per cent recovery was obtained. The lime present evidently prevented the zinc from floating.

Tests Nos. 4, 5, 7 and 8.—The reagents used at the company's plant were used. Mr. C. L. Dewar ran these tests and was unable to obtain satisfactory separation. Tests 4, 5 and 7 were run with water-gas tar from the Barrett Company. It was thought perhaps that the water-gas tar was the trouble, so a sample of the water-gas tar used at the company's mill was obtained and tried out in test No. 8. No improvement in the separation resulted.

Test No. 9.—This test was made to check the results obtained in tests Nos. 1, 2 and 3, and to see if the ore had altered in any way through oxidation since the first tests were made. When running tests on the North

¹Note particularly here that from experience on other ores, as well as this one, it has been found that the main bulk of the lime must be removed before floating the zinc, also that if sufficient lime is not added in the first place to insure the wetting effect on the iron remaining after decantation, the iron will tend to float with the zinc in the soda ash pulp.

²On the North Star ore equally good results were obtained without the addition of CuSO_4 .

Star ore it was observed that if the ore, ground to 30 mesh, stood for a couple of weeks, the lead oxidized and a good recovery could not be obtained without the addition of sodium sulphide.

The shipment of Sullivan ore had all been crushed to one-fourth inch for sampling. Only sufficient ore for three or four flotation tests was crushed to 30 mesh.

In test No. 9 a sample was taken from the one-fourth inch material and crushed to 30 mesh for the test. One thousand grammes of the freshly crushed ore was ground in the ball mill with 0.2 gramme Z cake, 5 lb. lime per ton and 1 lb. Selecto per ton for 30 minutes, to pass 200 mesh. The cell was frothed with Fumol No. 6. The procedure was the same as that used in the first three tests.

When the flotation of the lead was attempted, it was found that only a small quantity of concentrate was obtained. It was evident that the lead had become oxidized, so a little sodium sulphide was added to the pulp. After a short period of mixing, the lead was floated without much difficulty, but considerable iron also came up. Sodium sulphide has this effect as a rule. It was also expected that more iron would tend to float with the zinc in the subsequent flotation for that mineral. This was found to be the case when the zinc was floated, resulting in the production of a low-grade concentrate.

The results from test No. 9 show that the lead mineral in the ore has been altered in some manner and is difficult to float. The results on tests 4 to 9, therefore, cannot be accepted as reliable. In order to do any further work, in comparing the reagents suggested by the Consolidated Mining and Smelting Co., and the ones that have been found satisfactory by the Mines Branch laboratories, it will be necessary to obtain a fresh supply of ore.

Sullivan Lead-Zinc Ore

Head sample: Lead..... 11.40 per cent
Zinc..... 11.40 "
Iron..... 32.25 "

Test No.	Product	Weight grms.	Weight per cent	Pb. per cent	Zn. per cent	Per cent weight by per cent assay		Recover- ies		Remarks
						Pb.	Zn.	Pb.	Zn.	
1	Pb. conc...	133	13.7	52.60	6.1	706.9	83.6	63.0	7.5	1000 grms. —30-mesh ore ground for 30 mins. with 0.1 grm. Z cake, 2½ grm. lime and 1 lb/ton Selecto. Floated in Ruth with Fumol 3 drops (0.1 grm.) Tails dewatered and reloaded with 5 lb/ton soda ash and 5 lb/ton copper sulphate, 4 drops YZ and 1 drop Fumol. Zn. conc. re-treated making Zn. mids. and Zn. conc. (Fumol 2 drops).
	Pb. mids...	48	4.8	21.90	10.0	105.2	48.0	9.4	4.3	
	Zn. conc...	234	23.2	8.30	37.4	192.6	867.9	17.1	77.3	
	Zn. mids...	24	2.4	9.80	11.1	23.5	26.7	2.1	2.4	
	Tailing....	563	55.9	1.67	1.7	93.4	95.0	8.4	8.5	
2	Pb. conc...	85	8.5	63.18	5.4	537.1	45.9	48.2	4.0	1000 grms. —30-mesh ore ground for 30 mins. with 6 lb/ton lime and 1 lb/ton Selecto. Lead floated with 10 drops light creosote (hardwood) No. 25 (No. 31). Tails dewatered and Zn. floated with 6 lb/ton soda ash, 1 lb/ton copper sulphate, using No. 34 gravity fuel oil with No. 25 for froth.
	Pb. mids...	58	5.8	28.83	9.6	167.3	55.7	15.0	4.8	
	Zn. conc...	256	25.6	10.47	37.4	268.0	957.4	24.1	83.2	
	Zn. mids...	109	10.9	5.77	3.5	62.9	38.2	5.6	3.3	
	Tailing....	491	49.2	1.62	1.1	79.6	54.0	7.1	4.7	

Sullivan Lead-Zinc Ore—Continued

Test No.	Product	Weight grms.	Weight per cent	Pb. per cent	Zn. per cent	Percent weight by per cent assay		Recoveries		Remarks
						Pb.	Zn.	Pb.	Zn.	
3	Pb. conc...	122	12.1	52.38	4.9	638.7	59.8	55.1	5.2	1000 grms. —30-mesh ore ground in ball mill for 30 mins. with lime 5 lb/ton and Selecto 2 lb/ton. Run in Ruth using pine oil No. 5, coal tar creosote 0.03 grm. (Davidson Co.). Tailing dewatered and re-run for Zn. using No. 1 K.K. oil 6 drops. CuSO ₄ 1 lb/ton, Na ₂ CO ₃ 5 lb/ton.
	Pb. mids...	44	4.4	28.18	10.9	124.0	47.7	10.6	4.2	
	Zn. conc...	241	23.8	11.23	37.7	270.8	908.6	23.3	79.7	
	Zn. mids...	80	7.9	6.80	7.7	54.4	61.2	4.7	5.4	
	Tailing....	523	51.8	1.40	1.2	73.2	62.8	6.3	5.5	
4	Pb. conc...	105	10.5	53.12	11.5	557.0	120.7	50.2	10.4	1000 grms. —30-mesh ore ground for 30 mins. with soda ash 5 lb/ton, water-gas tar 0.3 lb/ton and coal tar creosote 0.3 lb/ton. Made lead conc. and lead middling CuSO ₄ 1 lb/ton and water-gas tar 0.2 lb/ton for zinc.
	Pb. mids...	97	9.7	27.20	13.53	264.0	131.4	23.7	11.4	
	Zn. conc...	196	19.6	8.13	21.65	159.3	424.3	14.3	36.8	
	Zn. mids...	242	24.2	3.90	0.25	94.4	223.8	8.4	19.5	
	Tailing....	360	36.0	1.52	7.00	37.8	252.0	3.4	21.9	
5	Pb. conc...	177	17.7	40.45	7.87	716.0	139.3	62.2	11.8	1000 grms. —30-mesh ore ground 30 mins. 5 lb/ton soda ash and 3 lb/ton water-gas tar and 3 lb/ton coal tar creosote to float lead. Zn. floated with a little water-gas tar and a drop of No. 350. CuSO ₄ 1 lb/ton was added. Lead tailing not dewatered before floating Zn.
	Pb. mids...	174	17.4	13.26	15.20	230.7	264.5	20.1	22.5	
	Zn. conc...	158	15.8	9.75	36.40	139.1	575.1	13.4	49.0	
	Zn. mids...	138	13.8	2.27	9.20	31.3	127.0	2.7	10.8	
	Tailing....	355	35.3	0.52	2.00	18.5	71.0	1.6	6.0	
6	Pb. conc...	147	14.6	44.28	11.20	64.5	16.35	58.8	13.8	1000 grms. —30-mesh ore ground 30 mins. with Z cake 0.1 grm., lime 5 lb/ton, soda ash 1 lb/ton. Floated Pb. by adding Fumol No. 6, 3 drops. Floated Zn. with CuSO ₄ 1 lb/ton and K.K. oil 3 drops, pine oil No. 5 1 drop. Pb. tailing was not dewatered before floating the zinc.
	Pb. mids...	160	15.9	13.88	12.60	22.1	20.04	20.2	17.0	
	Zn. conc...	47	4.7	14.03	26.03	6.6	12.4	6.1	10.4	
	Zn. mids...	42	4.2	7.74	7.38	3.3	3.1	3.0	2.6	
	Tailing....	610	60.6	2.17	10.96	13.15	66.4	11.9	56.2	
7	Pb. conc...	186	18.7	38.8	6.50	72.56	11.77	1000 grms. —30-mesh ore ground 45 mins. with soda ash 6 lb/ton, coal tar creosote 1 lb/ton, Barrett water-gas tar 1 lb/ton for lead. Zinc, CuSO ₄ 1.5 lb/ton, water-gas tar 6 drops. Pb. tails not dewatered before floating zinc.
	Pb. mids...	74	7.5	14.7	14.51	11.03	10.88	
	Zn. conc...	41	4.2	21.0	22.04	8.82	9.26	
	Zn. mids...	93	9.4	8.8	12.30	8.27	11.56	
	Tailing....	599	60.2	1.14	11.16	6.86	67.18	
8	Pb. conc...	190	19.0	36.43	11.12	69.22	21.12	1000 grms. —30-mesh ore ground for 30 mins. with soda ash 5 lb/ton. Consolidated water-gas tar 0.3 lb/ton. Coal tar creosote 0.3 lb/ton. To float zinc, CuSO ₄ 1 lb/ton, water-gas tar 0.2 lb/ton. Not dewatered between.
	Pb. mids...	129	12.8	10.78	15.55	13.82	9.90	
	Zn. conc...	56	5.6	23.20	15.94	12.99	8.92	
	Zn. mids...	105	10.4	7.59	14.61	7.89	15.20	
	Tailing....	525	52.2	0.76	9.94	3.97	51.90	
9	Pb. conc...	120.5	43.0	7.36	51.80	8.85	48.0	7.3	1000 grms. —30-mesh ore ground 30 mins. in ball mill with .2 gr. Z cake, 5 lb/ton lime and 2 lb/ton Selecto. Floated for lead in Ruth machine. Lead did not float well so Na ₂ S was added—came up but brought some iron with it. Dewatered lead tails and floated zinc adding 5 lb/ton soda ash, 1 lb/ton CuSO ₄ using No. 3½ gravity fuel oil.
	Pb. mids...	42.5	33.20	10.30	14.10	4.38	13.1	3.6	
	Zn. conc...	273.5	10.50	35.30	28.60	96.5	26.6	79.5	
	Zn. mids...	122	4.02	4.03	4.80	4.92	4.6	4.1	
	Tailing....	449	1.96	1.48	8.3	6.65	7.7	5.5	

GREEN TEST OF FLOTATION FEED

Heads -30 mesh ground 30 minutes in ball mill, washed on 200-mesh screen, oversize run on Ro-tap for 10 minutes:—

Mesh	Grammes.	Per cent weight	Per cent cum.
+100.....	1	0.1	0.1
+150.....	5	0.5	0.6
+200.....	59	5.9	6.5
-200.....	935	93.5	100.0

Report No. 188

FLOTATION TESTS ON COPPER MOUNTAIN ORE, COPPER MOUNTAIN, B.C.

R. K. Carnochan

Shipment.—A bag of ore, gross weight 89 pounds, was received July 18, 1922, at the Ore Dressing and Metallurgical Laboratories, from the Canada Copper Corporation, Allenby, B.C.

Purpose of experimental tests.—Test work was carried out on this ore to determine the results that would be obtained by different methods of treatment and to study the effect of different oil mixtures when used in the flotation of the ore.

Characteristics of the ore.—The ore is chalcopyrite and bornite disseminated through granite-diorite. There are also present small amounts of hematite, magnetite, and pyrite.

Sampling and analysis.—The lot of ore was crushed to -20 mesh and a sample for analysis was cut out and crushed to 100 mesh. This sample was found to contain:—

Gold.....	0.01 oz. per ton
Silver.....	0.26 "
Copper.....	2.15 per cent
Iron.....	3.45 "

EXPERIMENTAL TESTS

Test No. 3.—A lot of 1000 grammes of ore, -20 mesh, was ground with 3 grammes of lime in a small ball mill for 30 minutes. The ground ore was then floated in a small Ruth flotation machine with five drops TT mixture. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Percentage of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate.....	45.9	0.10	3.77	32.7	59.7	91.1	71.6
Middling.....	77.5	0.04	0.22	3.23	40.3	8.9	11.9
Tailing.....	885.0	tr.	tr.	0.39			16.5

A screen test on the flotation tailing gave:—

- 48+ 65 mesh.....	0.6 per cent
- 65+100 ".....	11.1 "
-100+150 ".....	11.9 "
-150+200 ".....	17.7 "
-200 ".....	55.7 "

Test No. 11.—A lot of 1000 grammes of ore, -20 mesh, was ground in a small ball mill for 30 minutes with 0.3 c.c. of heavy hardwood creosote and 0.2 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ground ore was then floated in a small Ruth flotation machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis			Percentage of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate.....	57.6	0.15	3.27	23.56	52.8	84.9	72.1
Middling.....	128.9	0.06	0.26	1.60	47.2	15.1	10.9
Tailing.....	819.4	tr.	tr.	0.39	17.0

Test No. 12.—A lot of 1000 grammes of ore was ground in a small ball mill for 30 minutes with 12 drops of water-gas tar. The ground ore was then floated with 3 drops of crude pine oil in a small Ruth machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate.....	32.8	33.82	58.1
Middling.....	83.5	4.55	19.9
Tailing.....	883.2	0.475	22.0

Conclusions from tests Nos. 3, 11, and 12.—In all of these three tests the concentrates are very good, but the tailings are all too high. This results in a low recovery of the copper. The screen test on the flotation tailing from test No. 3 shows that only 58.7 per cent of the tailing passes through 200 mesh, and that there is 0.6 per cent of the tailing +65 mesh. As the same amount of grinding was given to the ore used in all three tests, the tailings from tests Nos. 11 and 12 will be just as coarse as that of test No. 3. It was decided to make tests in which the ore would be ground finer, to determine whether this would improve the tailings.

Test No. 20.—A lot of 1000 grammes of ore, -200 mesh, was ground in a small ball mill for 30 minutes with 3 grammes of lime. The ground charge was then floated in a small Ruth machine with 8 drops TT mixture. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate.....	112.5	14.10	81.7
Middling.....	241.4	0.67	8.4
Tailing.....	643.9	0.30	9.9

Test No. 21.—A lot of 1000 grammes of ore, —200 mesh, was ground for 30 minutes in a small ball mill with 0.5 c.c. heavy hardwood creosote and 0.3 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ground ore was floated in a small Ruth machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate.....	186	9.00	85.7
Middling.....	291.5	0.50	7.5
Tailing.....	527.5	0.25	6.8

Test No. 23.—A lot of 1000 grammes of ore, —200 mesh, was ground in a small ball mill for 30 minutes with 5 grammes of lime and 10 drops of Barretts No. 634. The ground charge was floated in a small Ruth machine with 3 drops aldol. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate.....	146.1	10.78	83.2
Middling.....	195.2	0.71	7.3
Tailing.....	663.0	0.27	9.5

Conclusions from tests Nos. 20, 21, and 23.—The recoveries in these three tests are better than in tests Nos. 3, 11, and 12, but the grade of concentrates are poorer. The tailings are lower, but not as low as one might expect from grinding all to pass 200 mesh. It was thought that dry grinding all to pass 200 mesh and then grinding wet in the small ball mill might not be as good as all wet grinding. To determine whether this was true, it was decided to run a test in which the ore would be ground as in tests Nos. 3, 11, and 12, to make a screen test on the tailing, and to run each size from the screen test for copper.

Test No. 24.—A lot of 1000 grammes of ore, —20 mesh, was ground in a small ball mill for 30 minutes, and then floated in a small Ruth flotation machine with 8 drops of TT mixture, the concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate.....	61.9	25.50	79.4
Middling.....	125.4	1.52	9.6
Tailing.....	813.4	0.27	11.0

A screen test on the tailing and analysis for copper on the different sizes from the screen test gave:—

Mesh	Per cent weight	Per cent copper
— 48+ 65.....	0·8	0·85
— 65+100.....	10·7	0·62
—100+150.....	15·6	0·46
—150+200.....	15·0	0·34
—200.....	57·9	0·17

Conclusions from test No. 24.—This test seems to show that wet grinding to 200 mesh is better than dry grinding, as the —200 from the screen test runs 0·17 per cent copper, which is much lower than the tailings made in any of the tests, when the ore was all ground dry to —200 and then ground wet for 30 minutes in a ball mill. It was decided to make tests in which the ore would be ground wet, all to pass 200 mesh.

Test No. 25.—A lot of 1000 grammes of ore, —20 mesh, was ground wet for one hour in a small ball mill. The ground ore was screened wet on 200 mesh. The oversize was put back into the mill and ground for another hour. The ore from the mill was screened on 200 mesh, and the oversize was put back into the mill and ground for the third hour, at the end of which time it was all fine enough to pass the 200-mesh screen. The ground ore was partly dewatered by means of a suction filter and floated with 8 drops TT mixture in a small Ruth machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate.....	54·7	31·10	83·0
Middling.....	134·1	1·37	9·0
Tailing.....	818·2	0·20	8·0

Test No. 26.—A lot of 1000 grammes of ore, —20 mesh, was ground wet, all to pass 200 mesh in a manner similar to that described under test No. 25. The ground ore was partly dewatered by means of a suction filter, put in a small Ruth flotation machine, and 3 grammes of lime mixed in. The ore was then floated with 8 drops of TT mixture, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis per cent Cu.	Percentage of values Cu.
Concentrate.....	65·0	26·40	86·5
Middling.....	158·7	0·88	7·0
Tailing.....	756·0	0·17	6·5

Conclusions from tests Nos. 25 and 26.—Both these tests gave high-grade concentrates, low tailings, and good recoveries. If these tests are compared with tests Nos. 20, 21, and 23, it will be seen that tests Nos. 25 and 26 are much better than tests Nos. 20, 21, and 23. This shows that to prepare the ore for flotation wet grinding all to pass 200 mesh is much better than dry grinding to the same size.

GENERAL CONCLUSIONS

1. The ore can be successfully treated by flotation. About 90 per cent of the copper value of the ore could be recovered in a concentrate running about 26 per cent copper.

2. The ore would have to be ground wet to nearly all pass 200 mesh in order to secure a low tailing and a high recovery.

3. Any of the oil mixtures or reagents used in the tests given in this report would give good results on ore ground to nearly all pass 200 mesh, but of all the different oil mixtures or reagents used TT gave the best results.

4. The gold and silver values in the sample submitted are very low, and for this reason little work was done to determine how much of the gold and silver might be recovered by flotation. However, from the small amount of work done in this connexion, it would seem that the gold and silver are associated with the sulphide minerals, and when the ore is floated, the greater part of these values report with the copper minerals in the concentrate.

Report No. 190

CONCENTRATION OF A COPPER GOLD ORE FROM SURF INLET, B.C.

R. K. Carnochan

Shipment.—A sample of ore, gross weight 150 pounds, was received at the Ore Dressing and Metallurgical Laboratories on July 24, 1922, from the Belmont Surf Inlet Mines, Surf Inlet, B.C.

Purpose of experimental tests.—Test work was carried out on this ore to determine the results that would be obtained by different methods of treatment, and to study the effect of different oil mixtures when used in the flotation of the ore.

Characteristics of the ore.—The ore is white quartz with pyrite and chalcopyrite. No free gold or silver are visible to the eye.

Sampling and analysis.—The whole sample of ore received was crushed to one-fourth inch. A portion was cut from the main lot by means of a riffle sampler and ground to -20 mesh. A sample for analysis was cut from the -20-mesh material and ground to -100 mesh. This sample upon analysis showed the ore to contain:—

Gold.....	0.81 oz. per ton
Silver.....	0.23 “
Copper.....	0.425 per cent
Iron.....	5.74 “

EXPERIMENTAL TESTS

Flotation

Test No. 1.—1000 grammes of ore, -20 mesh, were ground in a small ball mill for half an hour with one pound of coal tar and coal tar creosote

per ton. The ground ore was then run in a small Ruth flotation machine making a concentrate and a tailing. The concentrate was re-run in the same machine making a concentrate and a middling.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate.....	56	6.20	1.48	4.53	59.1	49.6	58.9
Middling.....	61	2.46	0.94	1.62	25.6	34.3	23.0
Tailing.....	897	0.10	0.03	0.087	15.3	16.1	18.1

A screen test on the flotation tailings from test No. 1 gave the following:—

+ 65.....	0.1 per cent
— 65+100.....	1.0 "
—100+150.....	8.0 "
—150+200.....	26.7 "
—200.....	64.2 "

Test No. 2.—1000 grammes of ore, —20 mesh, were ground in a small ball mill for half an hour with 3 grammes of lime. The ground ore was then floated in a small Ruth machine with 5 drops of General Engineering Co.'s TT mixture. The concentrate produced was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate.....	14.5	22.55	4.17	20.62	62.1	40.7	72.4
Middling.....	49.6	3.08	1.40	1.73	29.0	46.6	20.8
Tailing.....	944.2	0.05	0.02	0.03	8.9	12.7	6.8

A screen test on the flotation tailings from test No. 2 gave:—

— 48+ 65.....	0.5 per cent
— 65+100.....	4.2 "
—100+150.....	16.8 "
—150+200.....	20.3 "
—200.....	58.2 "

Summary of Flotation Tests

1. Flotation gives very good concentrates but poor recoveries.
2. The TT mixture gave better results than coal tar and coal tar creosote in the above tests.

Tabling and Flotation Tests

Test No. 3.—Procedure:—A portion of the ore, 1200 grammes, was crushed to pass a 40-mesh screen and screened on 80 mesh. This gave:—

—40+80.....	442 grammes
—80.....	758 "

Each of the sizes was tabled separately on a small Wilfley table making a concentrate and tailing. The two concentrates were combined. The two tailings were also combined and then ground for 20 minutes in a small ball mill with 3 grammes of lime. After grinding, the tailings were floated in a small Ruth machine with 5 drops of TT mixture. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Table concentrate.....	135.8	1.37	0.32	2.24	37.7	56.4	67.1
Flotation concentrate.....	3.6	57.23	5.12	21.06	41.7	23.9	16.8
“ middling.....	31.6	2.56	0.48	2.00	16.4	19.7	13.9
“ tailing.....	916.4	0.02	trace	0.01	3.7	2.0
Loss.....	112.6	0.02	trace	0.01	0.5	0.2

A screen test on the flotation tailing from test No. 3 gave the following:—

— 48+ 65.....	0.8 per cent
— 65+100.....	4.4 “
—100+150.....	12.0 “
—150+200.....	11.2 “
—200.....	71.6 “

Test No. 4.—Procedure:—A sample of 1200 grammes of ore was crushed to —40 mesh and screened on 80 mesh:—

—40+80.....	463 grammes
—80.....	737 “

Each of the sizes was tabled separately on a small Wilfley table, making a concentrate and a tailing. The two concentrates were combined. The two tailings were combined and ground for 20 minutes with 0.6 c.c. of heavy hardwood creosote and 0.4 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ground tailing was then floated in a small Ruth machine. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Table concentrate.....	107.2	3.18	1.32	2.56	53.8	50.5	61.2
Flotation concentrate.....	69.6	4.00	1.60	2.18	44.0	39.8	33.9
“ middling.....	129.3	0.11	0.21	0.07	2.2	9.7	2.0
“ tailing.....	820.5	trace	trace	0.015	2.7
Loss.....	73.4	trace	trace	0.015	0.2

Test No. 5.—Procedure:—A sample of 1200 grammes of ore was ground to —40 mesh and screened on 80 mesh. This gave:—

—40+80.....	470 grammes
—80.....	730 “

Each of the above sizes was tabled on a small Wilfley table, making a concentrate and a tailing in each case. The two concentrates were combined. The two tailings were combined and ground in a small ball mill for 20 minutes with 12 drops of water-gas tar. The ground tailings were floated with 6 drops of crude pine oil in a small Ruth machine. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Table concentrate.....	115	2.76	1.52	2.48	52.6	65.0	63.6
Flotation concentrate.....	8.5	23.00	6.00	10.74	32.4	19.0	20.3
" middling.....	43.2	2.10	1.00	1.29	15.0	16.0	12.5
" tailing.....	972.2	trace	trace	0.015	3.4
Loss.....	61.1	trace	trace	0.015	0.2

SUMMARY OF TABLING AND FLOTATION

1. Tabling and flotation gave very good concentrates and very good recoveries. In test No. 4, which seems the best test made on the ore, the combined table and flotation concentrates assayed 3.56 oz. per ton gold, 1.43 oz. per ton silver, and 2.41 per cent copper. The recoveries in this test were: gold 97.8 per cent, silver 90.3 per cent and copper 95.1 per cent. These figures do not include any of the values in the middling product.

2. The various oils used in the above tests gave good results. TT mixture seems to have a better selective action for the copper, giving a higher grade concentrate. Heavy hardwood creosote, coal tar and coal tar creosote gave the highest recoveries, but with a much lower grade of concentrate.

SUMMARY OF EXPERIMENTAL TESTS

Tabling and flotation give much better results on the ore than flotation alone.

Conclusions.—The ore can be successfully treated by tabling and flotation. This method of treatment gives a very high recovery of the gold, silver and copper, and a high-grade concentrate. It is a difficult matter to determine from the small scale tests which reagents are the more economical to use for flotation. This point can only be definitely determined by plant operations over a period of time. It is also governed to a large extent by local and market conditions.

No tests were made, by flotation and table concentration, of the flotation tailing. This method should give equally as good results as tabling followed by flotation, and it would simplify plant operations to a considerable extent.

Report No. 191

GOLD ORE FROM CRANBERRY HEAD, NOVA SCOTIA (SHIPMENT No. 1)

C. S. Parsons

A shipment of gold ore was received February, 1923, from Mr. S. F. Johnson, 570 Columbus ave., Boston. The sample consisted of 280 pounds of ore, and was taken from a property located at Cranberry head, Yarmouth county, Nova Scotia.

Character of the ore.—The ore carries free gold in a quartz gangue and is associated with a small amount of arsenical pyrites, galena, and chalcopryrite. The values in the ore are very spotty. Nuggets of free gold are plainly visible in the quartz, and their size ranges from a maximum of 0.0328 inch in diameter to 0.0029 inch, and smaller, the greater portion being about 0.0082 inch. The grains of gold are very dense and show no porosity, their surface being smooth and the edges rounded. When the gold is broken free from the quartz a thin film of gold seems to remain attached to the surface of the quartz. The sample contained no wall rock.

Object of experimental work.—Tests were performed on the sample in order to obtain data for determining the best metallurgical treatment of the ore. The sample received is supposed to be a representative sample of the ore deposit.

ORE DRESSING AND METALLURGICAL TESTS

Sampling.—Great care had to be exercised when sampling the ore owing to the spotty distribution of the gold.

Head sample.—The assays of a number of carefully taken samples gave, gold 0.72 oz. per ton as the assay of this shipment.

Amalgamation and table tests.—These tests were run before the difficulty of sampling the ore was recognized. This accounts for the head samples of the ore used in the following tests containing less gold than the original sample from which they were cut.

Procedure of tests.—Four lots of ore were cut out from the shipment, and crushed respectively to 24, 30, 35, and 40 mesh. Each lot was amalgamated in a mercury-coated copper pan by panning. The tailing, from each after amalgamation, was tabled. These tests are numbered one to four, and are given in detail below.

Test No. 2

The ore was crushed through 30 mesh. A screen test of the amalgamation feed is given in table No. I*. The head sample assayed 0.57 oz. per ton. The recovery by amalgamation was 70 per cent. The amalgamation tailing was tabled, and the results are tabulated in table No. II below.

TABLE II

Product	Weight		Analysis		Per cent of gold values
	Grammes	Per cent	Au. oz.	As. per cent	
Concentrate.....	29	0.70	11.52	19.78	47.4
Middling.....	93	2.25	0.62	8.6
Tailing.....	3,870	93.00	0.08	44.0
Loss.....	168	4.05			
Total.....	4,160	100	0.18	100

*See p. 36.

RECAPITULATION

Recovery of gold by amalgamation.....	70.0 per cent
Recovery of gold by tabling, in table concentrate.....	16.4 "
Recovery of gold by amalgamation and in table concentrate....	86.7 "
Pounds table concentrate made from one ton of ore.....	14.0 pounds
Value of gold in one ton of table concentrate.....	\$230.40
Value of gold remaining in tailing, per ton.....	1.60

Test No. 3

The ore was crushed through 35 mesh. A screen test of the feed is given in table No. I. Head sample assayed 0.75 oz. per ton. The recovery by amalgamation was 70.5 per cent. The tailing from the amalgamation was tabled. The results are given below in table No. III. A screen analysis of the table tailing was made and is given in table No. IV.

TABLE III

Product	Weight		Analysis		Per cent of gold values
	Grammes	Per cent	Au. oz.	As. per cent	
Concentrate.....	31	0.74	12.62	16.6	45.7
Middling.....	139	3.30	0.37	6.1
Tailing.....	3,764	90.40	0.11	48.2
Loss.....	233	5.60
Total.....	4,167	100.0	0.22	100.0

RECAPITULATION

Recovery of gold by amalgamation.....	70.5 per cent
Recovery of gold in table concentrate.....	13.4 "
Recovery of gold by amalgamation and in table concentrate....	83.9 "
Pounds table concentrate from one ton of ore.....	14.8 pounds
Value of gold in one ton table concentrate.....	\$252.40
Value of gold remaining in tailing, per ton.....	2.20

TABLE IV

Mesh	Weight		Assay gold	Per cent content of gold	Per cent of gold total in table feed
	Grammes	Per cent			
-35+48.....	187	18.7	0.15	24.6	12.0
-48+65.....	224	22.4	0.12	23.5	11.6
-65+100.....	155	15.5	0.08	10.8	5.3
-100+150.....	79	7.9	0.07	4.8	2.3
-150+200.....	137	13.7	0.08	9.5	4.5
-200.....	218	21.8	0.14	26.7	13.2
Total.....	1,000	100.0	0.12	100.0	48.2

Screen test on table tailing, test No. 3.—This screen analysis shows that the gold in the coarse sizes was not freed by crushing. The +65-mesh sizes contain 48.1 per cent of the gold remaining in the tailing and the -200-mesh product 26.7 per cent. It is very evident that finer crushing is necessary to free the gold.

Test No. 4

The ore was crushed through 40 mesh. A screen test of the feed is given in table No. I. The head sample assayed 0.57 oz. per ton. The recovery by amalgamation was 65 per cent. The tailing from the amalgamation was tailed. The results are given below in table No. V.

TABLE V

Product	Weight		Analysis		Per cent of gold values
	Grammes	Per cent	Au. oz.	As. per cent	
Concentrate.....	26	0.6	14.26	20.85	45.4
Middling.....	66	1.6	0.64	5.2
Tailing.....	3,840	92.2	0.105	49.4
Loss.....	235	5.6			
Total.....	4,167	100.0	0.2	100.0

RECAPITULATION

Recovery of gold by amalgamation.....	65.0 per cent
Recovery of gold in table concentrate.....	15.7 "
Recovery of gold by amalgamation and in table concentrate....	80.7 "
Pounds table concentrate made from one ton of ore.....	12.0 pounds
Value of gold in one ton of table concentrate.....	\$285.20
Value of gold remaining in tailing, per ton.....	2.20

Test No. 9

A flotation test was made on the amalgamation tailing from test No. 4, which is -40-mesh material. The tailing without further grinding was mixed with the reagents and floated. The results are given in table No. VI. Head sample of flotation feed 0.20 oz. per ton.

TABLE VI

Product	Weight		Gold		Remarks
	Grammes	Per cent	Assay oz. per ton	Per cent of values	
Concentrate.....	73	7.3	1.66	59.2	Reagents used—coal tar and coal tar creosote, and pine oil. Amount 1 lb. per ton.
Tailing.....	929	92.7	0.09	40.8	
Total.....	1,002	100.0	0.205	100.0	

A screen test was made on the flotation tailing to determine the distribution of the gold remaining. The results are given below in table No. VII. The gold was found to be fairly evenly divided between all the sizes. It is evident that finer crushing is necessary to free the gold.

TABLE VII
SCREEN ANALYSIS FLOTATION TAILING

Mesh	Weight		Gold oz. per ton	Per cent of total gold
	Grammes	Per cent		
- 40+ 65.....	96.7	14.1	0.13	22.5
- 65+100.....	171.2	25.1	0.07	21.4
-100+200.....	202.5	29.6	0.06	21.8
-200.....	213.0	31.2	0.09	34.3
Total.....	683.4	100.0	0.082	100.0

Test No. 12

This test was made to determine the maximum percentage of gold recoverable by amalgamation when the ore was crushed to pass a 35-mesh Tyler standard screen.

Procedure.—2000 grammes of ore was crushed through 35 mesh and amalgamated in a jar. The pulp dilution used was 1 part of ore to 1 part water. 20 grammes or 1.0 per cent mercury was added. A handful of pebbles was thrown into the jar in order to keep the pulp from sticking to the sides. The addition of the pebbles caused some of the ore to be ground finer. After agitation for 3 hours, the mercury was separated from the tailing by panning.

TABLE VIII
SCREEN TEST OF SAMPLE BEFORE AMALGAMATION

Mesh	Weight		Gold		Accumulative per cent Au.
	Grammes	Per cent	Oz. per ton	Per cent of total	
- 35+ 48.....	231	23.1	0.80	25.5	25.5
- 48+ 65.....	168	16.8	0.80	18.5	44.0
- 65+100.....	156	15.6	0.87	18.7	62.7
-100+150.....	94	9.4	0.97	12.6	75.3
-150+200.....	104	10.4	0.84	12.1	87.4
-200.....	247	24.7	0.37	12.6	100.0
Total.....	1,000	100.0	0.725	100.0	

TABLE IX
SCREEN TEST ON TAILING AFTER AMALGAMATION

Mesh	Weight		Gold		Accumulative per cent Au.
	Grammes	Per cent	Oz. per ton	Per cent of total	
- 35+ 48.....	158.0	15.8	0.14	18.2	18.2
- 48+ 65.....	201.0	20.1	0.13	21.5	39.7
- 65+100.....	167.0	16.7	0.12	16.4	56.1
-100+150.....	92.5	9.2	0.11	8.4	64.5
-150+200.....	112.5	11.2	0.10	9.3	73.8
-200.....	265.5	26.5	0.12	26.2	100.0
Loss.....	3.5	0.3			
Total.....	1,000.0	100.0	0.12	100.0	

TABLE X
GOLD AMALGAMATED

Mesh	Per cent of gold amalgamated from each size
- 35+ 48.....	88.0
- 48+ 65.....	80.5
- 65+100.....	85.2
-100+150.....	88.85
-150+200.....	87.1
-200.....	65.1
Total.....	83.2

Explanation of the above tables.—Table No. VIII is a screen analysis of the ore before amalgamation, and gives the distribution of the gold values in the different sizes. For example: the figures in the first column, headed mesh, means the material passing through 35 mesh and remaining on 48 mesh. Column 2, headed weight grammes, gives the quantity of, say, -35+48 material in the sample. Column 3, under weight, is that quantity expressed in per cent of the total weight of the sample. Column 4, under gold, is the assay of that portion of the sample, say, passing through 35 mesh and remaining on 48 mesh, and is given in ounces per ton. Column 5 is that quantity of gold expressed in per cent of the total gold contained in the sample. Column 6, called accumulative per cent, is obtained by adding the figures obtained in column five.

Table No. IX is similar to table No. VIII.

Table No. X gives the amount of gold amalgamated from each size.

SUMMARY

Head sample assayed.....	0.73 oz. per ton
Amount of gold recovered by amalgamation.....	0.60 "
Amount of gold lost in tailing.....	0.13 "
Recovery by amalgamation.....	83.2 per cent

Discussion of above results.—Referring to and comparing the screen analyses on the amalgamation heads and tailing, it will be observed that the addition of the pebbles to jar produced a grinding action that reduced the quantity of material on the 48 and 65-mesh screen. This material caused a larger amount of gold to be amalgamated from these sizes than would otherwise have been the case. The figures for the quantity of gold which should actually amalgamate from the coarser sizes are therefore too high.

Test No. 13

AMALGAMATION AND CYANIDE TEST

This test was made with two objects in view. First, to determine the percentage of gold which would be recovered by a combined method of amalgamation and cyanidation from each respective size of a sample of the ore that had been sized on a series of screens given in table No. XI. Second, to obtain a comparison of the ratio of dissolution of the coarse gold in the 35, 48, and 65-mesh sizes with that of the fine gold on the 100, 150, and 200-mesh screens.

Procedure.—A 30-pound sample of ore was taken for the test and sized on the following screens, the weights being given in grammes (Table XI).

TABLE XI

Mesh	Weight		Assays		Per cent of total gold	Accumulative per cent total gold
	Grammes	Per cent	Gold oz. per ton	Arsenic per cent		
- 28+ 35.....	1,468	14.30	0.90	0.16	19.7	19.7
- 35+ 48.....	1,933	18.80	0.75	0.19	19.0	38.7
- 48+ 65.....	1,620	15.70	0.90	0.22	19.1	57.8
- 65+100.....	1,290	12.50	0.87	0.25	14.7	72.5
-100+150.....	806	7.80	0.87	0.16	9.2	81.7
-150+200.....	908	8.80	0.78	0.16	9.3	91.0
-200.....	2,277	22.10	0.30	0.22	9.0	100.0
Total.....	10,302	100.00	0.74	0.22	100.0	

Sampling.—It is extremely difficult to obtain correct samples of this ore, due to the spotty nature of the values, and to the coarse size of the gold particles.

Half of each of the above sizes down to the 100-mesh size had to be used as a sample to assay. Theoretically this amount is not sufficient, but in this case satisfactory checks were obtained.

Amalgamation tests.—The amount remaining after the samples were taken was used for the amalgamation tests. Each size was separately amalgamated in a revolving jar, a pulp of 1:1 being used, and with an amount of mercury equal in weight to 1 per cent of the ore in the charge. The contents of the jar were revolved for 2 hours. After amalgamation, the mercury was removed and dissolved in acid to recover the gold. The tailing was sampled and assayed for gold. The weight of gold obtained from the mercury plus the gold remaining in the tailing after amalgamation checked up very closely with assays obtained on the head sample of each of the sizes shown in table No. X. A recapitulation of these results is shown in table No. XII.

Discussion of results from amalgamation.—The tailings from the amalgamation of +200-mesh sizes were examined very carefully under the microscope. The gold remaining in the tailing was observed to be either attached to or enclosed in the sulphide and quartz particles. The amalgamation efficiency was practically 100 per cent on the free gold. In order to recover the remaining gold it would be necessary to crush each size finer. The curve shown in Fig. 1 shows very clearly that the recovery by amalgamation gradually increases as the sizes become smaller, until a maximum is reached at 150 and 200 mesh.

Referring to table No. XI, it will be observed that 57.8 per cent of the gold is in the sizes remaining on 65 mesh (that is coarser than 65 mesh) and that only 76.24 per cent of this gold is recoverable by amalgamation. In order to recover the remaining gold, this material would have to be re-crushed.

Cyanide tests.—Cyanide tests were made on the amalgamation tailing from each size. The object of these tests was to obtain a comparison

of the rate of dissolution of the coarse and fine gold, and the maximum extraction of the gold in each size. A 0.20 per cent cyanide (KCN) solution was chosen for the test. The amount of lime used was equivalent to five pounds per ton of ore, and the ratio of solution was 3:1. These conditions were kept constant in all tests, the time of treatment being the only factor varied.

Cranberry Head gold ore

TEST No. 13.

S. F. JOHNSON, Boston.

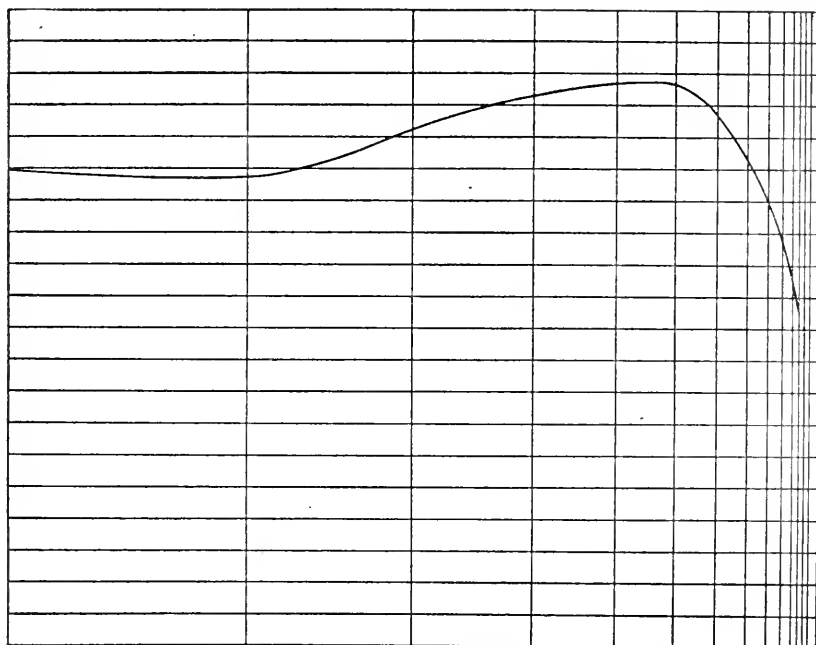


Fig. 1. Curve showing recovery by amalgamation on each size

The manner in which these tests were conducted was as follows: The amalgamation tailing from each screen product was divided equally into four lots. A lot from each size was given 1, 2, 4 and 5-days' treatment respectively. The third day period was skipped in every size, the reason being that it was desired to treat one sample for 5 days and there was only sufficient material on each size to make up four charges.

The results of the above tests are given in table No. XIV, and a series of curves from these figures plotted in Fig. 2. Table No. XIII gives the cyanide and alkali consumption for each test.

Discussion of cyanide results.—Referring to Fig. 2, it will be seen that in the -35+48-mesh material the rate of dissolution of the gold is rapid during the first 48 hours. At the end of the second day, the curve flattens out and very little increase in the extraction is obtained during the next 48 hours, and it is evident that this size material might be treated for an indefinite period without any appreciable increase in the extraction. It is

also evident that it would hardly be economical to carry the treatment beyond 72 hours. Careful examination of the amalgamation tailing from Cranberry Head gold ore.

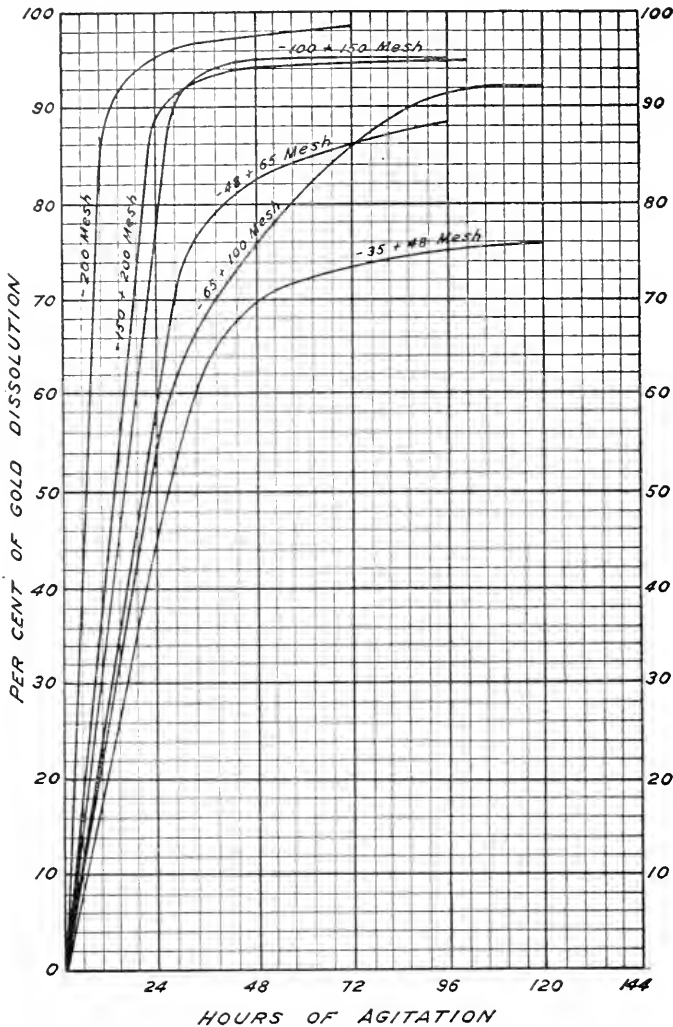


Fig. 2. Rate of dissolution of the gold in the different sized screen products obtained in equal intervals of time

this size, which constitutes the heads for the cyanide test, shows that no freed particles of gold were left unamalgamated in the tailing, but very thin coatings of gold could be seen on the surface of some of the quartz grains. The rapid extraction during the first 48 hours was, therefore, due to the dissolution of this gold. The flattening out of the curve in Fig. 2 is due to the inability of the cyanide to attack the gold still locked up in the quartz grains.

What has been said about the $-35+48$ -mesh material applies to the $-48+65$ mesh, and to some extent to the $+100$ -mesh material, but in both cases more gold is dissolved due to more surfaces being exposed by the reduction in size of the quartz grains.

In the $-100+150$ and $+200$ -mesh sizes, all the gold freed from the quartz grains was amalgamated, the remaining gold left for recovery by cyaniding being either attached to the surface of the quartz grains or entirely embedded in them. The condition of the gold in -200 -mesh product after amalgamation was not determined. The rate of dissolution of the gold was very rapid, and a maximum extraction was reached in the first 24 hours.

SUMMARY OF AMALGAMATION AND CYANIDE TREATMENT

1. By stamp milling the ore to 28 mesh and amalgamation on plates, a maximum extraction of 79.0 per cent of the gold is possible.
2. That in actual practice this result would be lower, probably about 70 per cent.
3. The fine gold does not amalgamate readily, but it is possible that the recovery might be increased by the addition of lime to the battery.
4. That there is no difficulty in obtaining a high extraction of the gold remaining in the amalgamation tailing by cyanidation.
5. That for maximum extraction by cyaniding, the amalgamation tailing should be re-crushed to pass 65 mesh.
6. That the rate of dissolution of the gold in the coarse sizes is so slow that it would not be practical to cyanide the gold by leaching methods.
7. That it is not necessary to continue cyanide treatment for a longer period than 48 hours.

CYANIDE TEST

Considerable detailed work was done on cyaniding this ore. A short synopsis of the findings is given under the following headings:—

Strength of solutions.—A number of tests were run using strengths of solutions varying from 0.05 to 0.25 per cent (KCN). The 0.05 per cent solution was found to give as high an extraction as the 0.25 per cent or stronger solution, in the same time of treatment. The consumption of cyanide was considerably lower with the 0.05 per cent solution. The consumption on the 0.05 per cent was approximately 0.6 pounds per ton, and with the 0.25 per cent solution 1.5 pounds per ton.

Alkalinity.—The alkalinity was found to have considerable influence on the cyanide consumption. If increased above a certain amount, more cyanide was consumed, and, if kept too low an increase in cyanide consumption was obtained. A protective alkalinity of about 0.02 per cent CaO should be maintained, and on the sample, 5 pounds of lime per ton of ore was sufficient to do this.

Time of treatment.—This depends entirely on the size to which the ore is crushed. The results from test No. 13 and the curves given in Fig. 2 show this very clearly.

Tests for Cyanide Consumption

Test No. 1.—To find if the ore contained water-soluble cyanicides, 1000 grammes ore was agitated for 24 hours with 600 c.c. water; 500 c.c. filtered off, 0.625 gramme KCN added, agitated 16 hours.

KCN found..... 0.124 per cent

No water soluble cyanide present.

Test No. 2.—To find if CaO increases the solubility of cyanicides. Same procedure as No. 1 except that 1 gramme CaO (20 pounds per ton) was added.

KCN found..... 0.124 per cent

CaO does not increase the solubility of the cyanicides.

Test No. 3.—To find the cyanide consumption when no lime is added, 100 grammes ore agitated 24 hours with 600 c.c. cyanide solution (0.125 per cent KCN).

KCN found..... 0.096 per cent
KCN consumed..... 3.48 pounds per ton
Gold in tailing..... 0.01 ounce per ton

Test No. 4.—To find if NaOH increases the solubility of cyanicides. Same procedure as No. 1 except that half a gramme of NaOH was added.

KCN found..... 0.125 per cent

NaOH does not increase the solubility of the cyanicides.

Test No. 5.—To find the effect of Na_2O_2 on the cyanide consumption. Same procedure as No. 3, except that 0.15 gramme Na_2O_2 was added (3 pounds per ton).

KCN found..... 0.112 per cent
KCN consumed..... 1.56 pounds per ton
Au. in tailing..... 0.010 ounce per ton

Na_2O_2 decreases the cyanide consumption.

Cyanide Test "A"

These are preliminary tests. The ore used in the tests was the -40-mesh amalgamation tailing from test No. 4.

Weight of ore taken, 200 grammes. Solution 600 grammes. Ratio 1:3.

Cyanide added		Cyanide recovered	
	gramme KCN		gramme
Making up added.....	0.3	Titrated 100 c.c. at .04 per cent KCN	= 0.04
Added 1st day.....	0.05	" 500 c.c. at .046 "	= 0.23
Total.....	0.35	Total, 600 c.c.....	= 0.27

Consumption $0.35 - 0.27 = 0.08$ gramme, that is 0.8 pound per ton of ore.

Average strength of solution used, 0.05 per cent KCN.

Lime added at start 0.5 gramme = 0.060 per cent CaO (lime 73 per cent CaO).

Lime in solution at end of 1st day = 0.035 per cent CaO.

" " 2nd day = 0.030 per cent CaO.

The tailing from the test assayed 0.068 oz. per ton.

The extraction was 66 per cent.

Total time of agitation was 44 hours.

Cyanide Test "B"

The same material was used as in test "A".

Weight of ore taken, 200 grammes. Solution used 600 grammes, Ratio 1:3.

Cyanide added		Cyanide recovered	
	gramme KCN		gramme
Making up added.....	0.6	Titrated 100 c.c. at 0.038 per cent.....	= 0.088
Added 1st day.....	0.06	" 500 c.c. at 0.094 per cent.....	= 0.470
Total.....	0.66	Total, 600 c.c.....	0.558

Consumption, $0.66 - 0.558 = 0.102$, that is 1.02 pounds per ton of ore.

Average strength of solution, 1.0 per cent KCN.

Lime added at start 0.5 gramme, 73 per cent CaO = 0.06 per cent CaO in solution.

Lime in solution the 1st day = 0.046 per cent CaO.

" 2nd day = 0.043 "

The tailing from the test assayed 0.07 oz. gold per ton.

The extraction was 65 per cent.

Time of agitation, 44 hours.

Cyanide Test "C"

Weight of ore used, 200 grammes. Solution, 600 grammes. Ratio, 1:3.

Strength of solution at start, 0.25 per cent KCN.

Cyanide consumption, 1.46 pounds per ton of ore.

Alkalinity maintained at end of test, 0.043 per cent CaO.

The tailing assayed 0.07 oz. gold per ton.

The extraction was 67 per cent.

Time of agitation, 44 hours.

A screen test was made on the cyanide tailing, with the following results:—

SCREEN TEST ON CYANIDE TAILING

Mesh	Weight		Gold			Per cent of total gold remaining in tailing
	Grammes	Per cent	Oz. per ton	Per cent	Accum. per cent	
— 35+ 48.....	8.1	4.15	0.072	4.5	4.5	1.58
— 48+ 65.....	36.7	18.80	0.095	27.2	31.7	8.90
— 65+100.....	40.4	20.70	0.101	31.7	63.4	10.45
—100+150.....	26.1	13.40	0.067	13.6	77.0	4.43
—150+200.....	25.7	13.20	0.047	9.4	86.4	3.10
—200.....	58.2	29.80	0.030	13.6	100.0	4.48
Total.....	195.2	100.0	0.066	100.0		

Test No. 14

Five samples of ore were cut out and crushed to pass 200 mesh. Head assay contained 0.72 oz. gold.

Test No. 1 was run with a 0.25 per cent KCN solution and with an alkalinity of 8 lbs. lime per ton.

Test No. 2 was run with a 0.10 per cent solution, and with an alkalinity of 8 lbs. lime per ton.

Test No. 3 was run with a 0.10 per cent KCN solution, and with an alkalinity of 5 lbs. lime per ton.

Test No. 4 was run with a 0.05 per cent KCN solution, and with an alkalinity of 8 lbs. lime per ton.

Test No. 5 was run with a 0.05 per cent KCN solution, and with an alkalinity of 5 lbs. lime per ton.

The results are given in table No. XV.

The tests were run for 24 hours each. The same extraction was obtained from each test, and amounted to 99.8 per cent of the gold.

It is evident that a 0.05 per cent KCN solution is as good as a 0.25 per cent KCN solution, and that if the alkalinity is increased above 5 pounds of lime per ton, the cyanide consumption is also increased.

TABLE I
SCREEN TESTS

<i>Test No. 1</i>			<i>Test No. 2</i>		
Mesh	Weight grammes	Per cent	Mesh	Weight grammes	Per cent
+ 20	0.0	0.0	+ 35	1.35	1.4
- 20+ 25	0.2	0.2	- 35+ 45	19.00	19.0
- 25+ 35	24.5	24.0	- 45+ 65	18.20	18.2
- 35+ 45	22.3	21.8	- 65+100	15.60	15.6
- 45+ 65	13.7	13.4	-100+150	10.60	10.6
- 65+100	10.7	10.5	-150+200	11.30	11.3
-100+150	7.0	6.9	-200	23.8	23.8
-150+200	7.4	7.3			
-200	16.2	15.9			

<i>Test No. 3</i>			<i>Test No. 4</i>		
Mesh	Weight grammes	Per cent	Mesh	Weight grammes	Per cent
+ 35	0.15	+ 35	0.3	0.3
- 35+ 45	13.50	13.5	- 35+ 45	6.05	6.0
- 45+ 65	19.6	19.6	- 45+ 65	19.30	19.3
- 65+100	17.25	17.2	- 65+100	19.40	19.4
-100+150	11.80	11.8	-100+150	12.50	12.5
-150+200	12.00	12.0	-150+200	13.70	13.7
-200	26.05	26.0	-200	28.95	28.9

TABLE XII
RECAPITULATION OF AMALGAMATION RESULTS
Test No. 13.

Product	Heads				Amalgamation tailing		Recovery gold per cent
	Weight grms.	Weight per cent	Calcul. assay oz. per ton	Per cent total gold	Gold oz. per ton	Per cent loss, gold	
- 25+ 35.....	1,468	14.3	0.90	17.7	0.23	25.5	74.5
- 35+ 45.....	1,933	18.8	0.75	19.5	0.20	26.7	73.3
- 45+ 65.....	1,620	15.7	0.90	19.6	0.17	18.9	81.1
- 65+100.....	1,290	12.5	0.87	15.1	0.12	13.7	86.3
-100+150.....	806	7.8	0.87	9.4	0.10	11.5	88.5
-150+200.....	908	8.8	0.78	9.5	0.09	11.5	88.5
-200.....	2,277	22.1	0.30	9.2	0.12	40.0	60.0
Total.....	10,302	100.0	0.72	100.0	21.3	78.7

TABLE XIII
TABLE OF KCN AND CaO CONSUMPTION

0.25 per cent KCN solution
0.060 per cent CaO = 5 lbs./ton lime

Test No. 13

Mesh	1st day 24 hours				2nd day 48 hours			4th day 96 hours			5th day 120 hours		
	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n	Lbs. per ton CaO cons.	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n
- 35+ 48	0.224	1.6	0.034	1.56	0.228	1.32	0.042	0.222	1.68	0.035	0.226	1.44	0.037
- 48+ 65	0.238	0.72	0.030	1.8	0.244	0.36	0.041	0.240	0.6	0.038			
- 65+ 100	0.234	1.0	0.030	1.8	0.224	1.56	0.031	0.230	1.2	0.028	0.226	1.44	0.030
-100+150	0.226	1.4	0.030	1.8	0.226	1.44	0.029	0.228	1.32	0.027			
-150+200	0.226	1.4	0.024	2.2	0.23	0.030	0.232	1.08	0.026	0.220	1.80	0.029
-200	0.230	1.2	0.021	2.3	0.212	0.029	0.226	1.44	0.023	0.228	1.32	0.020

TABLE IV
Test No. 13

Mesh	Heads assay	1st day 24 hours		2nd day 48 hours		4th day 96 hours		5th day 120 hours	
		Tailing assay	Extractn. per cent	Tailing assay	Extractn. per cent	Tailing assay	Extractn. per cent	Tailing assay	Extractn. per cent
- 28+ 35.....	0.23	This size was destroyed							
- 35+ 48.....	0.20	0.11	45	0.06	70	0.05	75	0.048	76
- 48+ 65.....	0.17	0.07	58	0.03	82.5	0.02	89		
- 65+ 100.....	0.12	0.05	55	0.03	76	0.01	92	0.01	92
-100+150.....	0.10	0.02	80	0.005	95	0.005	95		
-150+200.....	0.09	0.01	89	0.005	94	95	0.005	
-200.....	0.12	0.005	95	0.005	95	95	0.005	95

TABLE XV
Test No. 14

Test No.	At start						At finish						Amt. consumed	
	Per cent KCN soln.	Grm. KCN	KCN lbs. per ton	Per cent CaO soln.	Grm. lime	CaO lbs. per ton	Per cent KCN soln.	Grm. KCN	KCN lbs. per ton	Per cent CaO soln.	CaO lbs. per ton	KCN lbs. per ton	CaO lbs. per ton	
1	0.25	1.50	15	0.8	8	0.22	1.32	13.2	0.048	2.8	1.8	5	
2	0.10	0.60	6	1.1	8	0.084	0.504	5.04	0.047	2.8	0.96	5	
3	0.10	0.60	6	0.69	5	0.088	0.528	5.28	0.024	1.4	0.72	3.6	
4	0.05	0.30	3	1.1	8	0.036	0.216	2.16	0.045	2.7	0.84	5.3	
5	0.05	0.30	3	0.69	5	0.040	0.240	2.4	0.17	1.0	0.60	4	

Report No. 192

GOLD ORE FROM CRANBERRY HEAD, NOVA SCOTIA (SHIPMENT No. 2)

C. S. Parsons

Shipment.—A second shipment of gold ore was received May 2, 1923, from Mr. S. F. Johnson, 570 Columbus ave., Boston, U.S.A. The shipment consisted of 380 pounds and was taken from a property located at Cranberry head, Yarmouth co., Nova Scotia. This second shipment was submitted at the request of the Division as it was thought that the first shipment was not a representative sample of the ore.

Purpose of experimental work.—The object in conducting test work was to determine a metallurgical treatment for the ore.

Characteristics of the ore.—The ore of the second shipment as compared with that of shipment No. 1, contained considerable slate from the

walls of the vein. There was a larger percentage of arsenical pyrites and the gold was more finely disseminated. The slate contained some large crystals of arsenical pyrites.

Sampling and analysis.—Due to the spotty distribution of the gold in the ore great care was exercised in the sampling. The entire shipment was reduced to 10 mesh and cut in a riffle sampler to 93 pounds. This was further reduced to 20 mesh before cutting to 46.5 pounds. The samples for assaying were obtained from the 46.5 pounds by stage crushing and cutting until 500 grammes had been reduced to 150 mesh. The average assay of these samples gave 0.64 oz. gold per ton.

Experimental Tests.

Test No. 1A

The flow sheet as indicated by Fig. 3 was followed in this test. The ore was crushed to 30 mesh and amalgamated. The tailing from this first amalgamation was re-crushed to pass 100 mesh and again amalgamated. The tailing from the second amalgamation was passed over a Wilfley table and a concentrate containing arsenic and other sulphides was made and a final tailing was sent to waste.

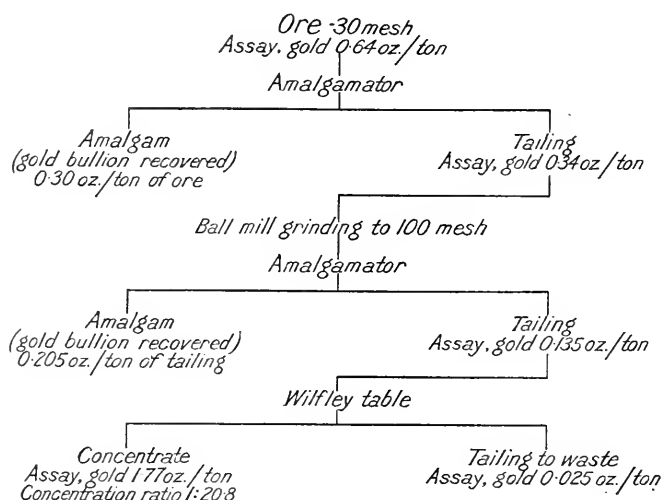


Fig. 3. Flow sheet used for test No. 1 A. Cranberry Head gold ore.

RECAPITULATION OF RESULTS

Assay of ore before treatment.....	0.64 oz. per ton
Tailing after first amalgamation.....	0.34 "
Tailing after second amalgamation.....	0.135 "
Tailing after tabling.....	0.025 "
Concentrate from table.....	1.77 "
Recovery of total gold by 1st amalgamation at 30 mesh.....	47.0 per cent
" " 2nd amalgamation at 100 mesh.....	32.0 "
" " table concentration.....	16.7 "
Recovery of total gold.....	95.7 "

The table concentrate represents 4.8 per cent by weight of the ore, that is, for every 100 tons of ore treated, 4.8 tons of concentrate will be obtained, assaying approximately 1.77 oz. gold per ton.

The above figures are a recapitulation of the results of the test which is given in detail below.

A 500-gramme sample was cut out, having first been crushed to pass 30 mesh, and a screen analysis made to determine the distribution of the gold values before amalgamation.

TABLE I
SCREEN ANALYSIS TEST NO. 1A

Mesh	Weight		Gold	Per cent of total gold
	Grammes	Per cent	Assay oz. per ton	
- 30+100.....	838.2	55.9	0.665	59.9
-100+150.....	111.8	7.5	0.65	7.8
-150+200.....	159.0	10.6	0.96	16.4
-200.....	389.2	26.0	0.38	15.9

This material was amalgamated by placing in a jar with 1 per cent by weight of mercury and rotating for one hour. A pulp consistency of one part of water to one of ore was used. The mercury was separated by panning and retorted. The amount of gold amalgamated as above stated was found to be 47.0 per cent of the gold in the ore. The gold recovery was 0.30 oz. per ton. The tailing was assayed and found to contain 0.34 oz. per ton of gold. A screen analysis was made on this tailing to determine the distribution of the gold.

TABLE II
SCREEN ANALYSIS OF 1ST AMALGAMATION TAILING

Mesh	Weight		Gold	Per cent of total gold
	Grammes	Per cent	Assay oz. per ton	
- 30+ 48.....	371.8	30.0	0.43	41.0
- 48+ 65.....	190.3	15.5	0.38	18.7
- 65+100.....	159.8	12.9	0.32	13.0
-100+150.....	94.5	7.5	0.225	5.4
-150+200.....	114.5	9.2	0.235	6.9
-200.....	308.5	24.9	0.19	15.0

TABLE III
DISTRIBUTION OF GOLD BEFORE AND AFTER 1ST AMALGAMATION

Mesh	Before amalgamation	After amalgamation	Gold recovered from each	
	Oz. per ton	Oz. per ton	Oz. per ton	Per cent
- 30+100.....	0.665	0.398	0.267	40.1
-100+150.....	0.650	0.225	0.425	65.4
-150+200.....	0.960	0.235	0.725	75.5
-200.....	0.380	0.190	0.190	50.0

It is very evident from table No. III that the gold is not all freed in the coarser sizes, and that in order to obtain a high recovery by amalgamation it will be necessary to crush the ore to 100 mesh at least. The sample was crushed dry. Wet crushing, as will be shown later, gives much higher recoveries of gold.

Second amalgamation.—The second step in this test was to re-crush a portion of the first amalgamation tailing to 100 mesh and amalgamate. This lot was crushed dry in rolls and a disk crusher. The amalgamation was proceeded with as in the first part of the test. The tailing assayed 0.135 oz. of gold per ton. This is equivalent to a recovery of 60.3 per cent of the gold remaining in the first amalgamation tailing.

Tabling.—The second amalgamation tailing was concentrated on a small Wilfley table. A concentrate and tailing were made. The concentrate represented 4.8 per cent of the weight of the ore, and assayed 1.77 oz. per ton gold. The tailing was low in gold, assaying only 0.025 oz. per ton, equivalent to a value of 50 cents per ton. The table recovered 78 per cent of the gold remaining in the second amalgamation tailing. A total recovery of 95.7 per cent of the gold was made by this method of treatment.

Test No. 1B

Purpose of test.—The purpose of this test was to determine the relative merits of two methods of treatment, namely, double amalgamation followed by tabling of second tailing as in test No. 1A, and the method

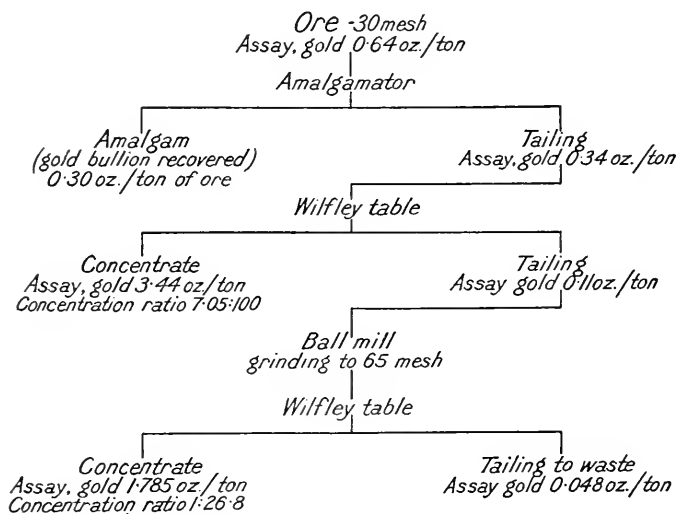


Fig. 4. Flow sheet used for test No. 1 B. Cranberry Head gold ore.

described below. The flow sheet as indicated by Fig. 4 was followed in this test. Half of the tailing from the first amalgamation test No. 1A was used in this test. This tailing was concentrated without further crushing,

on a Wilfley table, and a concentrate and tailing produced. The tailing from this first tabling was crushed wet in a ball mill to pass 65 mesh, and again tabled. A recapitulation of the results follows:—

Assay of heads before treatment.....	0.64 oz. per ton
“ tailing after first amalgamation.....	0.34 “
“ concentrate from first tabling at 30 mesh.....	3.44 “
“ tailing “ “ 30 “	0.11 “
“ concentrate “ second “ 65 “	1.785 “
“ tailing “ “ 65 “	0.048 “

The ratio of concentration was 7.05 : 100 in the first tailing.
 “ “ 3.74 : 100 in the second tailing.

Total gold in heads recovered by amalgamation at 30 mesh.....	42.07 per cent
“ “ “ in 1st table concentrate at 30 mesh....	37.50 “
“ “ “ 2nd “ “ 65 “ ...	10.40 “
	94.90 “

Detailed description of test.—Part of the tailings from the first amalgamation at 30 mesh in test No. 1A was used in this test. This tailing which assayed 0.34 oz. per ton was concentrated on a Wilfley table, two products being made, a concentrate and tailing.

TABLE IV

Product	Weight		Gold	Per cent of gold values
	Grammes	Per cent	Assay oz. per ton	
Concentrate.....	164.7	7.05	3.44	70.4
Tailing.....	2,171.0	92.95	0.11	29.6

A screen analysis of the first table tailing is given in table No. V. It is very evident from these figures that the gold is not freed from the coarser particles of gangue, and that crushing the whole to pass 100 mesh would probably do this sufficiently.

TABLE V
SCREEN ANALYSIS

Mesh	Weight		First table tailing		Per cent Au. recovered from each size	Per cent of total gold
	Grammes.	Per cent	Assay oz. per ton	Assay x weight per cent		
— 30+ 48.....	328.8	30.7	0.16	62.8	44.1
— 48+ 65.....	172.5	16.1	0.125	67.1	18.0
— 65+100.....	137.4	12.8	0.085	73.4	9.9
— 100+150.....	85.7	8.1	0.054	76.0	3.6
— 150+200.....	87.9	8.2	0.090	61.6	6.3
— 200.....	257.0	24.0	0.085	55.2	18.1

The tailing from the primary table which contained 0.11 oz. gold was re-crushed in a ball mill to pass 65 mesh and again concentrated on a Wilfley table. The result of this operation is given in table No. VI:—

TABLE VI

Product	Weight		Gold	Per cent of gold values
	Grammes	Per cent	Assay oz. per ton	
Concentrate.....	41.2	3.7	1.785	58.9
Tailing.....	1,060.5	96.3	0.048	41.1

TABLE VII
SCREEN ANALYSIS SECOND TABLE TAILING

Mesh	Weight		Gold	Per cent of gold values
	Grammes	Per cent	Assay oz. per ton	
+200.....	374.0	47.7	0.03	29.2
-200.....	410.3	52.3	0.065	70.8

TABLE VIII
DISTRIBUTION OF GOLD BEFORE AND AFTER PRIMARY TAILING

Mesh	Gold, oz. per ton		Per cent of values recovered
	First amalgamation tailing	Primary table tailing	
- 30+ 48.....	0.43	0.160	62.8
- 48+ 65.....	0.38	0.125	67.1
- 65+100.....	0.32	0.085	73.4
-100+150.....	0.225	0.054	76.0
-150+200.....	0.235	0.090	61.7
-200.....	0.190	0.085	55.2

Discussion of results of tests Nos. 1A and 1B.—The recoveries were practically the same in both tests. Comparing the results of the two tests, it is evident that the treatment used in test No. 1A offers a number of advantages over the methods used in test No. 1B. By the double amalgamation, over 79 per cent of the gold was recovered as bullion, while in test No. 1B, only 47 per cent was obtained as bullion. The costs of operation of the two methods would be practically the same. The first cost of the machinery used in the flow sheet of test No. 1B would be greater than that used in test No. 1A on account of twice the number of tables being used. The chief advantage of the first flow sheet over the second is that 79 per cent against 47 per cent of the gold was obtained as bullion ready to market. In the second flow sheet 47.9 per cent of the gold is in a concentrate associated with refractory material, compared with only 16.7 per cent in the first flow sheet.

Pan amalgamation tests on table concentrates.—The concentrate obtained from the first tabling in test No. 1B was used for this test. The concentrate weighing 164.7 grammes was placed in a mortar with 10 per cent of its weight of mercury. Water was added to make a thick pasty pulp which would hold up the mercury and the mixture was ground for four hours. The pulp was gradually diluted, the grinding being continued for another hour until the concentrate had been all reduced to a slime. The slime was decanted and washed away and the mercury was collected and retorted to recover the gold. The following results were obtained:—

Amount of gold in concentrate before amalgamation.....	3.44 ozs. per ton
Amount of gold in concentrate after amalgamation.....	0.63 “
Amount of gold recovered as bullion.....	2.81 “

This means that 81.7 per cent of the gold in the concentrate was recovered as bullion ready for the market. The residue remaining contained 0.63 oz. gold per ton, valued at, roughly, \$12.60, and 2.07 per cent arsenic. This concentrate could probably be sold to smelters in the United States owing to the favourable situation of the mine, which is understood to be on the sea coast.

AMALGAMATION TESTS

A series of amalgamation tests were run to determine the best size to crush the ore for the primary amalgamation. It had previously been observed that the recovery fell off if the ore was crushed finer than 35 mesh. The following tests were run:—

TABLE IX

Test No.	Size to which ore was crushed for amalgamation	Percentage of gold recovered by amalgamation as bullion
	mesh	per cent
3.....	—20	47.3
4.....	—28	48.9
5.....	—35	59.4
2.....	—50	47.5

The above table shows that the best size to crush to for the primary amalgamation is between 30 and 40 mesh.

LARGER SCALE TEST

A larger scale test was made on 10,545 grammes of ore, using flow sheet Figure 5. The ore was crushed wet in a ball mill to pass 40 mesh and then passed over amalgamation plates. The amalgamation tailing was then re-crushed to 100 mesh and passed over a second series of amalgamation plates. The second amalgamation tailing was concentrated on a Wilfley table and a concentrate and a final tailing were made. Recapitulation of results:—

Assay of ore before treatment.....	0.64 oz. per ton
Assay of tailing after first amalgamation.....	0.18 “
Assay of tailing after second amalgamation.....	0.05 “
Assay of table tailing.....	0.035 “
Assay of table concentrate.....	1.33 “
Per cent of total gold recovered by amalgamation at 40 mesh..	71.9
Per cent of total gold recovered by amalgamation at 100 mesh..	20.3
Per cent of total gold recovered by amalgamation.....	92.2
Per cent of total gold recovered by table concentration.....	2.6
Per cent of total gold recovered by combined operations.....	94.8

The table concentrate represented 6.34 per cent of the weight of the original ore. That is for every 100 tons of ore treated 6.34 tons of concentrate would be obtained assaying 1.33 ozs. per ton.

Corduroy blankets were used after the amalgamation plates in the second amalgamation at 100 mesh. The material remaining on the blankets was passed over the table with the second amalgamation tailing, hence the increase in amount of gold values shown in the table concentrate.

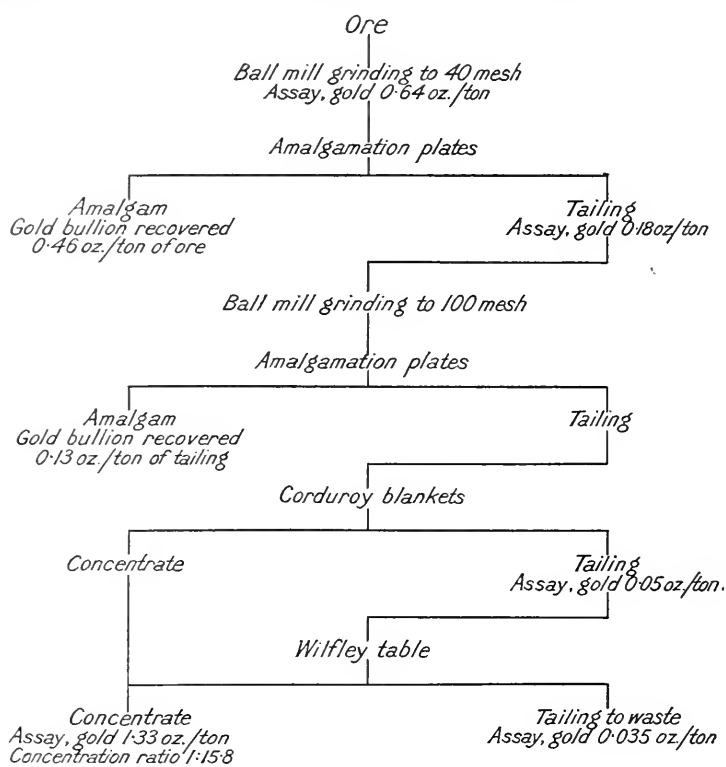


Fig. 5. Flow sheet used for larger scale tests, Cranberry Head gold ore.

Roasting and pan amalgamation of table concentrate.—An assay ton of the concentrate was roasted and the residue amalgamated by pan amalgamation, the idea being that the arsenic could be recovered in the form of a crude arsenical oxide by roasting, and the residue amalgamated for the recovery of the gold values. The results were as follows:—

Concentrate before roasting.....	1.33 oz. per ton
Concentrate after roasting.....	0.96 "
Loss in roasting.....	0.37 "
Loss in residue after amalgamation.....	0.23 "
Recovered by amalgamation.....	0.73 "
Recovered.....	55 per cent

This method of treating the table concentrate was not very successful.

CONCLUSIONS AND RECOMMENDATIONS

It is understood that the property from which these samples were taken has not been developed to any extent, and that no tonnage of ore has been blocked out. The samples received were taken from the surface, as a considerable portion of the rock showed surface weathering. Taking into consideration that the property is still in the development or prospect stage, the flow sheet as indicated by Figure 6 is recommended for the recovery of the gold.

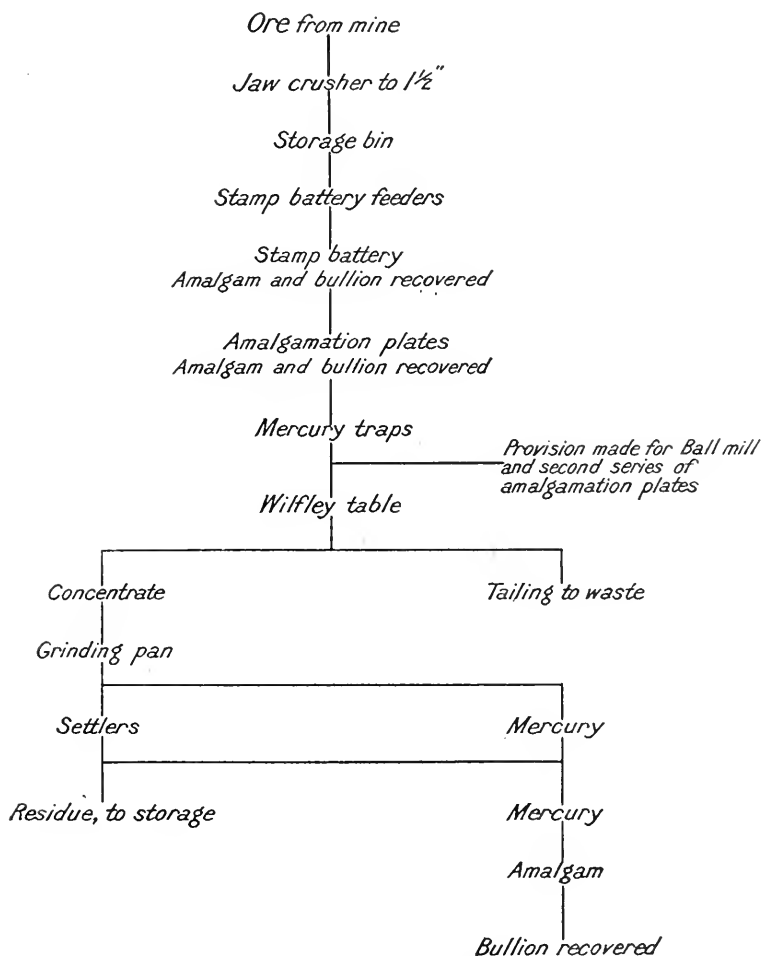


Fig. 6. Flow sheet recommended for treatment of Cranberry Head gold ore.

The flow sheet can be added to and improved later when sufficient ore has been developed in the mine to warrant a larger expenditure. It is advisable to install a 1000-pound ten-stamp battery of the California type with a wide and deep mortar for inside amalgamation. The mortar should

be at least 18 inches wide at a 6-inch discharge. For the most satisfactory operation the battery should be fed by a feeder, of the Challenge type. A jaw crusher should be used to crush the ore to $1\frac{1}{2}$ inches before it is fed to the battery. The plate area should be fairly large, at least 25 square feet per stamp, and the plates should be stepped. A good type of mercury trap could be used at the end of the plates. The tailings from the stamps should be run over one Wilfley table for concentration. Provision should be made to treat the table concentrate, and for this purpose the use of a grinding pan with a capacity of 2 tons of concentrate per day, together with mercury settlers to recover the mercury, is recommended.

The results of the foregoing tests indicate that the table concentrate will contain 1.3 or more ounces of gold per ton, and that an 80 per cent recovery of this gold can be obtained by fine grinding and amalgamation in a pan.

When the mill is built, provision should be made for the installation of a ball mill for regrinding the tailing from the battery plates, and for further amalgamation on stationary plates, so that if later it is found advisable to regrind, this equipment can be installed with the minimum of expense.

Report No. 193

AMALGAMATION AND CYANIDE TESTS ON THE ORE FROM THE OPHIR LODE MINE, BRITISH COLUMBIA

R. K. Carnochan

A shipment of ore, gross weight 20 pounds, was received May 7, 1923, from Dr. H. B. Morrison, Nelson, B.C. This ore came from the Ophir Lode mine, situated on the divide between Gainer creek, a fork of the Lardeau river, and Marsh Adams creek, a fork of the Westfall river.

The ore contains values in gold and silver. The sample received was very much weathered and contained a considerable quantity of iron sulphide.

Tests were desired on the sample to determine if amalgamating at 80 mesh would be better than amalgamating at 40 mesh, and if it would be necessary to cyanide after amalgamating.

Head sample.—The ore was crushed to 40 mesh, some metallica being obtained. A sample for assay was cut out and ground to pass 80 mesh, more metallica being obtained. The -80 mesh material and the two lots of metallica were assayed with the following result:—

	Au. oz. per ton	Ag. oz. per ton
+40 mesh metallica.....	0.33	0.06
+80 mesh metallica.....	0.80	0.03
-80 mesh.....	1.76	0.46
Heads.....	2.94	0.60

Amalgamation.—Amalgamation tests were made on the ore by mixing in a small pebble jar for three hours, about 1000 grammes of ore, 400 c.c. of water, and 100 grammes of mercury. After mixing, the mill charge

was put through a hydraulic classifier to separate the mercury from the tailings. The tailing was then dried, weighed, and sampled for assay. Three tests were made on ore ground to -40 mesh, and three on ore ground to -80 mesh. The results were:—

	Au. oz. per ton	Ag. oz. per ton
-40 amalgamation tailing.....	0.35	0.24
-40 amalgamation tailing.....	0.36	0.23
-40 amalgamation tailing.....	0.28	0.15
Average.....	0.33	0.21
-80 amalgamation tailing.....	0.38	0.20
-80 amalgamation tailing.....	0.28	0.16
-80 amalgamation tailing.....	0.33	0.21
Average.....	0.33	0.19

Amalgamation at 40 mesh gave in metallics and amalgam a gold recovery of 88.8 per cent and a silver recovery of 65.0 per cent. Amalgamation at 80 mesh gave in metallics and amalgam a gold recovery of 88.8 per cent and a silver recovery of 68.3 per cent.

Cyanidation.—Cyanidation tests were made on all amalgamation tailings. About 200 grammes of tailing was taken for each test, a 24 hour agitating period was used, and the tailings were ground to -100 and -200 mesh. The average results obtained are:—

	Au. oz. per ton	Ag. oz. per ton
-40 amalgamation tailing cyanided at -100 mesh.....	0.01	0.03
-40 amalgamation tailing cyanided at -200 mesh.....	0.02	0.01
-80 amalgamation tailing cyanided at -100 mesh.....	0.00	0.05
-80 amalgamation tailing cyanided at -200 mesh.....	0.03	0.02

The above table shows that the lowest tailings in gold are apparently secured by cyaniding at -100 mesh. This is not the case, however, as cyaniding at -200 mesh will always give as good, and usually better, results than at -100. The tests show that cyaniding at either -100 or -200 will give a tailing running about 0.02 oz. gold and 0.03 oz. silver per ton. With this tailing, the recoveries made on the ore by amalgamating and cyaniding are, gold 99.3 per cent, silver 95.0 per cent.

The average cyanide consumption in the tests was 5.5 pounds per ton of amalgamation tailing treated, and the average lime consumption was 7.2 pounds per ton.

CONCLUSIONS

1. Amalgamation followed by cyanidation is a suitable method of treating the ore submitted, as the recoveries are very high, and the tailings very low.

2. Amalgamation at 40 mesh is preferable to amalgamation at 80 mesh, as the gold recovery at 40 mesh is the same as that at 80 mesh; the silver recovery is only slightly lower, and the mercury does not flower to nearly the same extent.

3. Cyanidation will be necessary after amalgamation, as the increase in recovery certainly warrants it, and the amalgamation tailings are too high to discard.

4. For cyaniding, the amalgamation tailings will need to be ground to 100 mesh. Finer grinding than 100 mesh is not necessary.

5. The consumption of cyanide is fairly high, probably due to the oxidized condition of the ore. It would be expected on fresh unoxidized ore that this consumption would be much less.

Report No. 194

THE CONCENTRATION OF THE COMPLEX SULPHIDE ORE OF THE EUSTIS MINE, EUSTIS, QUEBEC

C. S. Parsons

Shipment.—A carload shipment of about 20 tons of ore was received May 29, 1923, from the Eustis Mining Co., Ltd., Eustis, Que. The shipment was submitted as representative of the company's milling ore from the Eustis mine.

Purpose of experimental tests.—The object in conducting experimental test work was to determine a satisfactory and commercial method of separating the minerals, with the production of marketable copper concentrates with high recoveries of the values in the ore.

Previous experimental tests.—The company had, previous to making this shipment, carried on small laboratory tests for a year under the direction of Mr. J. H. Allen, of Knox and Allen, consulting metallurgists of New York. The results of these tests were at first erratic, but as the work progressed, more uniform and very encouraging results were obtained from the small scale laboratory tests.

Request for tonnage check tests.—In order to eliminate any possibility of failure, the company made application to the Department to have tonnage check tests conducted to verify the results of the small scale tests, before proceeding with the alterations to their concentrating plant.

Arrangements for tonnage check tests.—It was arranged that Mr. Allen, representing the company, should be present during the running of the tests and that the work should be carried on under the joint supervision of Mr. Allen and an official of the Department.

Previous work of a similar nature.—The production of a high-grade copper concentrate from complex sulphide ores has, in many cases, presented difficult problems, and it is only in the last few years that successful solutions have been obtained commercially. The Department has given considerable attention to these problems, and has obtained encouraging results on a number of Canadian ores by differential flotation. Experimental work on such ores as the Flinflon, Hidden Creek, and the Sudbury copper-nickel ores indicated that the chalcopyrite could be floated selectively from the iron sulphides by maintaining an alkaline pulp with lime.

Decision as to experimental work.—It was desired by the company that the work should include two particular lines of investigation—

First: The flotation of the chalcopyrite from the other minerals of the ore with the production of a marketable copper concentrate (at least 20 per cent copper) with high recoveries of the copper content, and,

Second: The flotation of the iron sulphide from the gangue with the production of a marketable concentrate of high sulphur content.

This report deals entirely with the differential flotation of the chalcopyrite from the iron sulphides and gangue minerals. The flotation of the pyrite will be taken up later under a separate investigation.

Characteristics of the ore.—The ore is a complex sulphide consisting of chalcopyrite associated with pyrite and small quantities of calcite, quartz, and other gangue minerals. The shipment received contained about 3 per cent copper. The chalcopyrite is finely disseminated throughout the pyrite, the sulphides constituting 80 per cent of the ore, the remaining 20 per cent being made up of gangue minerals. A careful examination of the ore indicated that it should be ground to pass 100 mesh to free the chalcopyrite.

Sampling and analysis.—Each run was sampled separately. Three tons were weighed out, crushed in a jaw crusher and rolls to one-quarter inch. A tenth part was cut out by a Vezin automatic sampler and further reduced for analysis. An approximate analysis of the shipment would be:—

Copper.....	2.9 per cent
Iron.....	40.0 "
Sulphur.....	45.0 "
Insoluble.....	7.5 "
Other gangue.....	4.6 "

EXPERIMENTAL TESTS

Small scale tests by the Department.—Before the large scale tests were run, a number of small scale tests were run in order to gain some idea of the characteristics displayed by the ore during flotation, such as the nature of the froth, concentrate, tailing, etc.

Summary of small scale tests.—A summary of the results of these tests is given in table No. I.

TABLE I
SMALL SCALE TESTS

Test No.	Product	Analysis per cent Cu.	Remarks
1	Concentrate..... Middling..... Tailing.....	25.2 6.3 0.59	1000 grammes ore —80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.
2	Concentrate..... Middling..... Tailing.....	29.11 4.78 0.855	1000 grammes ore —80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.
3	Concentrate.....	18.1	1000 grammes ore —80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.
4	Concentrate.....	29.69	1000 grammes ore —80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.

Conclusions from small scale tests.—These tests clearly indicated that a high-grade concentrate could be produced. A low tailing, however, was not obtained. The poor recovery was probably due to the high dilution of the pulp used in these tests, which was approximately 1:4. When running the large scale tests, it was found that in order to obtain a low tailing, the pulp must be kept fairly dense, approximately 1:2.5 in the cells.

Large scale tests.—The same flow sheet was followed in all tests. The one-fourth-inch material was fed to a 4½-ft. Hardinge ball mill carrying a mixed ball charge of 4,000 pounds of 2-inch and 1½-inch balls. The mill was operated in closed circuit with a Dorr classifier, the overflow of the classifier going to Callow flat bottom cells. The rate of feed was maintained at about 1,000 pounds per hour. The results of only the first three tests are given in condensed form from the notes kept by Mr. Allen.

Test No. 1

Synopsis of run.—This test was started with a clean circuit, the ball mill being empty. Lime was added to the ball mill with the ore at the rate of 5 pounds per ton. When the ball mill started to discharge it was noticed that the pulp did not have the right appearance for flotation. The pulp was discharging fairly thick from the mill; more water was added, and the mill discharge reduced to 50 per cent solids. The appearance of the pulp improved slightly. By this time the pulp had commenced to enter the flotation cells, but conditions were not right as the froth was pasty and dark looking, and not lifting much mineral. The lime was gradually increased to 10 pounds per ton before correct conditions were obtained, the ball mill discharge being maintained at 50 per cent solids. The condition of the pulp discharging from the mill began to improve and a bright float of copper was obtained in the classifier. The character of the froth in the cells started to show a radical change, becoming brighter and lively with fast breaking small bubbles which surrounded larger slower breaking ones. Copper began to colour the froth heavily at the feed end of the cell, while the froth near the discharge end became white and watery. It was known from previous experience that the right conditions had been obtained.

After obtaining this correct condition, attention was paid to the oils. Barretts No. 634 oil was fed to the ball mill, and the General Engineering Company's YZ reagent was added to the classifier overflow to the cells.

Density samples were taken every 15 minutes on the ball mill discharge and the classifier overflow, and every 30 minutes on the cell discharge.

Samples for analysis.—No samples were taken until near the end of the run, as the chief object in this first test was control.

Analysis of samples.—A sample of the concentrate and tailing from the cells was taken and analysed:—

TABLE II
ANALYSIS OF SAMPLES—TEST NO. 1

Product	Cu. per cent	Fe. per cent	S. per cent	Insol per cent
Head sample.....	2.90	40.6	46.4	7.10
Concentrate.....	23.66			
Tailing.....	0.49			

Tests Nos. 2 and 3

Procedure.—The same procedure was followed as in the previous test.

Oils and reagents used.—The amount of lime was kept constant at 10 pounds per ton. The oils and soluble flotation reagents were varied. The best results were obtained with Barretts No. 634 and TT mixture of the General Engineering Co.

Samples.—Separate samples were taken during the use of each reagent, the samples being changed every time the reagents were changed. One continuous sample was, however, taken of the concentrate and tailing during the whole run. The analysis of these samples is given in table No. III.

TABLE III
ANALYSIS OF SAMPLES—TESTS NOS 2 AND 3

Product	Cu per cent	Remarks
Head sample, test No. 2.....	2.95	
Concentrate No. 1, " 2.....	24.42	
Concentrate No. 2, " 2.....	24.60	
Concentrate No. 3, " 2.....	19.58	
Tailing No. 2, " 2.....	0.55	
Final tailing, " 2.....	1.20	Taken during whole run.
Head sample, " 3.....	2.88	
Final concentrate, " 3.....	22.30	Taken during whole run.
Concentrate No. 1, " 3.....	21.16	
Tailing No. 3, " 3.....	0.20	
Tailing No. 4, " 3.....	0.45	
Final tailing, " 3.....	0.33	Taken during whole run.

TABLE IV
SCREEN TEST ON FINAL TAILING SAMPLE—TEST NO. 2

Size	Weight grammes	Per cent
— 65+100.....	21.1	4.2
—100+150.....	43.5	8.7
—150+200.....	194.5	38.8
—200.....	241.5	48.2

TABLE V
SCREEN TEST ON FINAL TAILING SAMPLE—TEST NO. 3

Size	Weight grammes	Per cent
— 65+100.....	5.2	1.0
—100+150.....	24.0	4.8
—150+200.....	102.8	20.6
—200.....	368.0	73.6

Test No. 4

Procedure.—Similar to the preceding tests.

Synopsis of run.—The test was started at 10.10 a.m. with a built-up circuit remaining from the previous run. The rate of feed was approximately 1,000 pounds per hour.

Oils and reagents used 10.10 a.m. to 2.00 p.m.:—

Barretts No. 634.....	54 drops per minute
General Eng. Co., YZ.....	15 " "
No. 5 steam distilled pine oil.....	4 " "
Lime.....	10 pounds per ton

The pine oil was varied slightly from time to time to regulate the quantity of froth in the cells.

Temperature of pulp ball mill discharge.—The pulp at 47 per cent solids was 90°F.

Sample No. 1, from 11.00 a.m. to 2.00 p.m.—Time of running on sample No. 1 was three hours.

Analysis:

Heads.....	3.00 per cent copper
Concentrate.....	24.70 " "
Tailing.....	0.29 " "

Pulp dilutions, 11.10 a.m. to 2.00 p.m.—The ball mill discharge varied from 47 per cent solids at the start to 55 per cent at 1.10 p.m., with an average of 29 per cent solids. The classifier overflow varied from 46 per cent solids at the start to 29 per cent at 1.45 p.m. The average range was 29 to 36 per cent, with an average density of 33 per cent solids.

The tailing from the cells ranged during the greater part of the time between 22 and 26 per cent solids with an average of about 24 per cent. The density samples on the ball mill and tailing from cells were taken every half hour, and the classifier overflow every fifteen minutes.

Oils and reagents used, from 2.00 p.m. to 4.45 p.m.:—

Barretts No. 634.....	54 drops per minute
General Eng. Co., TT.....	30 " "
No. 5 pine oil.....	1 " "
Lime.....	10 lbs. per ton

Sample No. 2, from 2.00 p.m. to 4.45 p.m.—The time of running on this sample was 2.75 hours.

Analysis:

Heads.....	3.00 per cent copper
Concentrate.....	22.77 " "
Tailing.....	0.12 " "

Pulp dilution from 2.00 p.m. to 4.45 p.m.—The ball mill discharge ranged between 50 to 55 per cent solids, with an average of 53 per cent. The classifier overflow varied between 29 and 31 per cent solids, with an average of 30 per cent. The tailing from the cells varied between 24 and 30 per cent, with an average of 27 per cent.

TABLE VI
SCREEN TEST ON FLOTATION TAILING—TEST NO. 4

Mesh	Weight grammes	Per cent
+ 65.....		
— 65+100.....	19.5	3.9
— 100+150.....	31.7	6.34
— 150+200.....	68.8	13.76
— 200.....	380.0	76.00

Summary:—

Started test at.....	10.10 a.m.
Completed test at.....	4.30 p.m.
Running time.....	6 hrs. 20 mins.
Rate of feed.....	952 pounds per hour

Analysis of samples taken continuously from beginning to end of test:—

Heads.....	3.00 per cent copper
Concentrate.....	23.00 " "
Tailing.....	0.24 " "

Conclusions.—The density of the feed to the flotation cells was much higher and more constant in the last half of the test than during the first half, but owing to the change from YZ to TT reagents, it is impossible to determine definitely whether the lower tailing obtained during the latter part of the run was due to the more uniform and higher density of the pulp in the cells or to the change of reagents. The opinion of the writer is that the lower tailing was due to the more uniform and higher density of pulp. The reason for this is based on the observation that when the density of the pulp was the highest, the appearance of the cell indicated that the best separation was being obtained.

Test No. 5

Procedure.—Similar to previous test, with the exception that 1,500 pounds of balls were removed from the ball mill, leaving 2,500 pounds.

Synopsis of run.—The test was started at 10.15 a.m. with a built-up circuit remaining from previous run. The rate of feed was maintained at approximately 1,000 pounds per hour.

Oils and reagents used at start:—

Barretts No. 634.....	34 drops per minute
General Eng. Co., TT.....	30 " "
No. 5 pine oil.....	1 " "
Lime.....	10 lbs. per ton

Conditions of froth at start.—The froth was pasty and dead at the start, but gradually cleaned up as the alkaline circuit was built up.

Change of reagents.—The Barretts No. 634 was increased to 70 drops, and the TT mixture reduced to 18 drops. A change in the character of the froth was noted, indicating over oiling and the lifting of more iron sulphide. The Barretts oil was reduced to 54 drops per minute and the TT raised to 30 drops. At the time of starting No. 1 sample the TT was increased to 36 drops per minute.

Sample No. 1, from 11.40 a.m. to 12.30 p.m.—The time of running on this sample was 55 minutes.

Analysis:

Heads.....	2.98 per cent copper
Concentrate.....	19.10 " "
Tailing.....	0.22 " "

Pulp dilution between 11.40 a.m. and 12.30 p.m.—The ball mill discharge varied between the narrow limits of 53 to 54 per cent solids. The classifier overflow density ranged between the two limits of 27 and 34 per cent solids, with an average of 28 per cent. The tailing discharge from the cells averaged 28 per cent.

Change of samples at 12.30 p.m.—The cells were working better and the concentrate looked cleaner.

Oils and reagents, 12.30 p.m. to 2.30 p.m.—These remained as before,—

Barretts No. 634.....	54 drops per minute
General Eng. Co., TT.....	36 " "
Pine oil.....	1 " "
Lime.....	10 lbs. per ton

Sample No. 2, 12.30 p.m. to 2.30 p.m.—A total time of two hours.

Analysis:

Heads.....	2.98 per cent copper
Concentrate.....	19.50 " "
Tailing.....	0.14 " "

Pulp dilution, 12.30 p.m. to 2.30 p.m.—Ball mill discharge varied between 49 and 55 per cent solids, with an average of 51. The classifier overflow varied between 26 and 29 per cent solids, remaining constant the greater part of the time at 26 per cent solids. The tailing discharge from the cells averaged 28 per cent solids.

Change of operating conditions at 2.30 p.m.—The feed, instead of all passing through one rougher cell and being returned through the second rougher cell, was divided, half going to each cell. The tailing from each rougher cell was sampled separately.

Oils and reagents, 2.30 p.m. to 4.45 p.m.—The reagents and oils were not altered.

Sample No. 3, 2.30 p.m. to 4.45 p.m.—The tailing from the two rougher cells were marked A and B respectively:—

Analysis:

Heads.....	2.98 per cent copper
Concentrate.....	20.20 " "
Tailing A.....	0.15 " "
Tailing B.....	0.12 " "

Pulp dilution, 2.30 p.m. to 4.45 p.m.—The ball mill discharge varied between 51 and 58 per cent solids. The classifier overflow varied between 29 and 33 per cent solids. The greater part of the time the density remained constant at 29 per cent solids. The tailing from the cells averaged 28 per cent solids.

Summary:

Total ore fed.....	6,000 pounds
Total time of running.....	6.5 hours
Rate of feed.....	.923 pounds per hour

Analysis of samples taken continuously from start to end of test:—

Heads.....	2.98 per cent copper
Concentrate.....	19.00 " "
Tailing.....	0.20 " "

Screen test on flotation tailing.—This screen test is given in table No. VII.

TABLE VII
SCREEN TEST FLOTATION TAILING—TEST No. 5

Mesh	Weight grammes	Per cent
+ 65.....		
— 65+100.....	6.5	1.3
— 100+150.....	25.0	5.0
— 150+200.....	98.5	19.7
— 200.....	370.0	74.0

It will be observed that the removal of 1,500 pounds of balls from the mill did not affect the crushing to any extent. The material —200-mesh was only 2 per cent less than in the previous test, while the amount remaining on 100 mesh was considerably less than when more balls were used in the mill. This was due in all probability to the more careful regulation of the classifier.

Conclusions.—The use of TT mixture in place of YZ mixture, which is difficult to obtain, gave satisfactory results.

FINAL SUMMARY AND CONCLUSIONS OF ALL TEST WORK

1. If the shipment submitted was representative of the ore to be mined, the results of the experimental work justify the erection of a mill, the flow sheet of which should conform to the procedure used in conducting the tonnage check tests.

2. The results of the experimental work show that a high-grade concentrate of at least 20 per cent copper, with a recovery of better than 90 per cent, can be expected.

3. That the ore should be crushed to pass 100 mesh, and so that 80 per cent will pass 200 mesh.

4. That an alkaline pulp, made so by lime, is essential to obtain selective flotation of the copper from the iron.

5. That when the mill is first started, Barretts No. 634 and TT mixture should be used until in perfect operation, then other and cheaper oils can be tried.

6. That the temperature of the pulp in the ball mill should be maintained as low as possible, not exceeding 90° F.

7. That a high pulp density should be maintained in the flotation cells. Samples taken from the rougher cell discharge should show a density of not less than 26 per cent solids.

8. That the pulp should be delivered direct to the flotation cells from the grinding circuit. No attempt should be made to thicken this feed by dewatering. The efficiency of the closed grinding circuit should, if necessary, be sacrificed to obtain this condition. The classifier overflow density should be maintained at 30 per cent solids.

9. That the lime should be added ahead of the ball mill, as long contact is necessary.

10. That one cleaning operation is all that is necessary.

11. That to obtain efficient operation, conditions must be kept uniform. This is very clearly indicated from the tests conducted. The pulp densities must be maintained constant. The amount of lime added should be controlled by chemical determinations of the alkalinity of the tailing water from the cells.

Report No. 195

EXPERIMENTAL TESTS ON GOLD ORE FROM KIRKLAND LAKE GOLD MINING CO., LTD., KIRKLAND LAKE, ONTARIO

R. K. Carnochan

Shipment.—Two samples of gold ore and one of mill tailings were received on December 22, 1922, at the Ore Dressing and Metallurgical Laboratories, from the Kirkland Lake Gold Mining Co., Ltd., Kirkland Lake, Ont. The samples as received were numbered 1, 2, and 3, and weighed 95, 60, and 32 pounds respectively.

Purpose of experimental tests.—In the milling of their ore, the company was not securing a very low tailing, and tests were desired on the samples to determine if anything could be done to improve the extraction.

Characteristics of the ore.—Sample No. 1 was high-grade ore containing molybdenite. The gangue of this ore was mostly red porphyry with some quartz. A small amount of sulphides was present as minute specks. The molybdenite occurs as a very thin coat in the fracture faces. No free gold was visible. Sample No. 2 was run-of-mine containing no visible molybdenite, but otherwise similar to sample No. 1. Sample No. 3 was mill tailing.

Sampling and analysis.—After selecting some specimens from lots Nos. 1 and 2, they were crushed to one-fourth inch. A small portion was cut out of each and crushed to 40 mesh. Samples for assay and analysis were cut out of the 40-mesh material and also out of sample No. 3. Assays and analyses were made on all three samples with the following results:—

	Sample No. 1	Sample No. 2	Sample No. 3
Gold.....	4.33 oz./ton.	0.12 oz./ton.	0.04 oz./ton.
Silver.....	0.55 "	0.04 "	0.05 "
Molybdenite.....	0.30 per cent	0.15 per cent	0.10 per cent
Iron.....	4.20 "		
Silica.....	53.60 "		
Sulphur.....	2.30 "		
Tellurium.....	0.10 "		
Alumina.....	17.15 "		
Lime and magnesia.....	6.00 "		
Carbon.....	Nil		
Loss on ignition.....	3.50 "		

Due perhaps to the presence of tellurium in the ore, difficulty was found in assaying both the samples received and the products from the test work. The figures given above for samples Nos. 1, 2, and 3 are quite reliable.

Tests on Sample No. 1

Cyaniding.—Samples of the ore were ground to -100, -150, and -200 mesh. Portions of each size were agitated for different lengths of time in a solution made up to titrate 0.25 per cent KCN and 0.10 per cent CaO by using sodium cyanide and soda ash. The following table shows the

results of these tests. Unless otherwise stated, agitation was effected in a bottle closed by a rubber stopper.

Time of agitation hours	Tails -100		Tails -150		Tails -200		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
8	0.06	0.04	0.10	0.08	0.12	0.14	
8	0.19	0.08	0.13	0.09	0.12	0.09	
16	0.10	0.10	0.12	0.12	0.04	0.04	
16	0.20	0.12	0.16	0.12	0.12	0.11	
24	0.18	0.16	0.16	0.18	0.08	0.10	
24	0.16	0.08	0.13	0.06	0.12	0.17	
24	0.09	0.03	Agitation by revolving stirrer in a pail.
24	0.14	0.05	Agitation in open bottle revolving horizontally.
24	0.09	1 lb/ton sodium peroxide used.
24	0.09	0.14	3 per cent H ₂ SO ₄ wash, then water wash before cyaniding.
48	tr.	Agitation by revolving stirrer in a pail.
48	0.07	

The above table shows:—

1. The difficulty of securing checks in assaying.
2. The ore can be readily cyanided to 0.20 oz. or less gold per ton. This means a recovery of 95.4 per cent of the gold. This is a good recovery on an ore running \$89.50 in gold, but the tailing runs \$4.13 and is too high to throw away.
3. Fine grinding, finer than -100 mesh, does not greatly lower the tailing.
4. Increasing the time of agitation beyond 8 hours does not lower the tailing to any great extent.
5. The same results were obtained in the test work when different methods of agitation were used.
6. Sodium peroxide does not improve the cyaniding of the ore.
7. A preliminary acid wash does not make the ore more amenable to cyanidation.

Flotation and cyanidation.—In view of the fact that cyanidation alone did not produce a tailing low enough to discard, it was decided to make some tests to determine if flotation used in conjunction with the cyanide process would lower the tailing to a suitable point. In making the tests a small laboratory Ruth flotation machine was used for the flotation work. In cyaniding, a solution titrating 0.25 per cent KCN and 0.10 per cent CaO was used. Agitation was effected in bottles closed by rubber stoppers.

—	Flot. concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Floating at -200 mesh with pine oil and coal oil and then cyaniding for 24 hours.....	65.93	5.67	0.05	
Floating at -200 mesh with TT mixture and then cyaniding for 24 hours.....	52.68	3.50	0.12	0.19
Floating with heavy hardwood creosote, coal tar and coal tar creosote at -200 mesh and then cyaniding for 24 hours.....	16.50	2.19	0.09	0.18
Cyaniding at -200 mesh for 24 hours, and then floating with heavy hardwood creosote, coal tar and coal tar creosote.....	0.33	0.47	0.16	0.15
Cyaniding at -200 mesh for 24 hours and then floating with TT mixture.....	tr.	0.17	tr.	0.31

The above table shows:—

1. The difficulty of securing a representative sample of the ore. Apparently the sample used in the last test shown in the table did not run as high as the average of sample No. 1, for as both the flotation concentrate and tailing run a trace in gold, cyaniding for 24 hours at -200 mesh must have reduced the ore to a trace in gold. This does not agree with the tests described under cyaniding, so it would seem that the sample taken for the test was not representative.

2. Flotation used with cyaniding improves the recovery somewhat, but does not give a tailing low enough to discard.

Cyaniding and Tabling.

—	Table concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Cyaniding at -200 mesh for 24 hours and then tabling the cyanide tailing.....	1.25	1.65	0.10	0.12

This test shows that tabling improves the extraction somewhat but does not give a low enough tailing to discard.

Conclusions, Sample No. 1

1. It is easy to make a high recovery of the gold in this sample by cyanidation.

2. When used with cyanidation, tabling and flotation both improve the extraction.

3. It has been found impossible to secure a very low tailing.

Tests on Sample No. 2

Cyaniding.—Cyaniding tests were made on sample No. 2. These tests were made in the same manner as the cyaniding tests on sample No. 1.

Time hours	Tails -100		Tails -200		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
24 48	0.05	0.15	tr. tr.	tr.	Agitation by revolving stirrer in pail.

The above table shows:—

1. The difficulty of securing a representative sample, the tails in the first tests being much too high.

2. Fine grinding and cyaniding for 24 hours gives a high recovery and a low tailing.

Flotation and tabling.—As sample No. 2 was low-grade ore, it was thought advisable to see what results could be obtained by floating and tabling, as this would cost less than cyaniding, and if it worked would give a larger margin of profit per ton.

	Flotation concentrate		Table concentrate		Table tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Floating at -100 mesh with heavy hardwood creosote, coal tar and coal tar creosote and tabling flotation tailing.....	tr.	tr.	0.50	0.28	tr.	tr.
Floating at -65 mesh with heavy hardwood creosote, coal tar and coal tar creosote and tabling flotation tailing.....	tr.	tr.	0.70	0.43	tr.	tr.
Floating at -80 with Barretts No. 634 and sulphuric acid and tabling flotation tailing.	1.03	0.14	1.56	0.71	0.02	0.05
Floating at -80 with Barretts No. 634 and sulphuric acid and tabling flotation tailing.	1.10	0.33	0.60	0.34	tr.	tr.
Ore at -20 ground in ball mill to about -65 then floated with Barretts No. 634 and sulphuric acid and flotation tailing tabled..	0.48	0.69	0.28	0.29	tr.	tr.

The above table shows:—

1. Floating and tabling give very good results on sample No. 2.

2. Barretts No. 634 and sulphuric acid are suitable reagents to float with, much better than the creosote mixture used in the first two tests, as this mixture did not float any values.

3. It is a question which would be the better method to use on a low-grade ore like sample No. 2. Flotation and tabling would cost less than cyaniding, but there would result a concentrate which would have to be treated at the mill, as it is scarcely valuable enough to ship to a smelter.

Tests on the concentrate produced by floating and tabling show that it can be cyanided to 0.06 oz. gold per ton. This means that the recovery by floating and tabling, and cyaniding the concentrates would be just about the same as that obtained by straight cyaniding, so it would seem that as far as can be determined by the small scale tests conducted, floating and tabling followed by cyaniding the concentrates would be the better method to treat low-grade ore such as represented by sample No. 2.

Conclusions, Sample No. 2

1. Cyaniding gives a high recovery and a low tailing.
2. Floating and tabling, followed by cyaniding the concentrate gives a high recovery and a low tailing.

Tests on Sample No. 3

Sample No. 3 was taken from the cyanide tailings of the Kirkland Lake Gold Mining Company's mill. This sample was slightly damp when received, but it was not dried, as it has been found that even the slight heating necessary to dry a tailing will make it more amenable to cyanidation. A small portion of sample No. 3 was sized and each size assayed with the following results:—

Size	Per cent by weight	Au. oz. per ton	Ag. oz. per ton
+ 65.....	4.7	0.09	0.10
— 65+100.....	25.7	0.04	0.05
— 100+150.....	19.8	0.02	trace
— 150+200.....	22.4	0.03	0.04
— 200.....	27.4	0.05	0.05

Cyaniding.—Cyaniding tests were made on sample No. 3. These tests were made in the same manner as the cyaniding tests on sample No. 1.

Time hours	Tails not ground		Tails — 100		Tails — 200		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
24 48	trace	trace	trace	trace	trace	trace	Agitation by revolving stirrer in pail.
	trace	trace	

The above table shows:—

1. Sample No. 3 is readily cyanided to a trace in gold.
2. The sample did not need to be reground to secure a low tailing.

Conclusions, Sample No. 3

The Kirkland Lake Gold Mining Co. are having difficulty in securing a low tailing, and numerous tests on their mill tailing show that it is impossible to reduce it to a trace in gold by straight cyaniding. In con-

sideration of the fact that the company has made a large number of tests on many samples, and that the Division has had only one sample of tailing to experiment with, their results should be regarded as more reliable and the conclusion drawn, that the sample received by the Division was not representative, or that the air drying it received made it amenable to cyanidation.

CONCLUSIONS ON SAMPLES SUBMITTED

It is very regrettable that none of the samples received corresponded to the ore now being milled by the Kirkland Lake Gold Mining Company. Sample No. 1 is much higher than their ordinary run-of-mine, and sample No. 2 much lower. However, from the tests made on the samples submitted, it would seem that similar ore carrying a moderate value in gold could be cyanided with good results.

The molybdenite in the ore seems to have no deleterious effect on the cyanide process.

Report No. 196

EXPERIMENTAL TESTS ON GOLD ORE FROM WRIGHT-HARGREAVES MINES, LTD., KIRKLAND LAKE, ONTARIO

R. K. Carnochan

Shipment.—A box of ore, gross weight 146 pounds, was received on February 1, 1923, at the Ore Dressing and Metallurgical Laboratories, from the Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ont.

Purpose of experimental tests.—Tests were desired on the sample submitted, to determine what results the different methods of milling would give, and to see whether the molybdenite present in the ore interfered with the cyanide process.

Characteristics of the ore.—The ore consists of red porphyry and dark quartz intimately mixed, and carrying a small amount of sulphides, mostly chalcopyrite and molybdenite. The molybdenite occurs in a thin film on slickensided faces.

Sampling and analysis.—After the selection of suitable specimens the balance of the shipment was crushed to one-quarter inch. A portion was cut out by means of a riffle sampler from the $-\frac{1}{4}$ -inch material and crushed to pass a 20-mesh screen. A sample was cut from the -20 -mesh material and ground to -100 mesh for analysis and assay. This sample was found to contain on analysis:—

Gold.....	5.62 oz. per ton	Sulphur.....	1.16 per cent
Silver.....	0.54 "	Tellurium.....	0.04 "
Molybdenite.....	0.32 per cent	Alumina.....	7.49 "
Iron.....	2.86 "	Lime and magnesia...	2.34 "
Silica.....	81.10 "	Carbon.....	trace
		Loss on ignition.....	1.80 per cent

Experimental Tests

Cyaniding.—A series of cyaniding tests were made on the ore ground to -100 , -150 , and -200 mesh. A solution made up to titrate 0.25 per cent KCN and 0.10 per cent CaO by using sodium cyanide and soda ash

was used in all the tests. Different times of agitation were used. Unless otherwise stated, agitation was secured by the use of rubber stoppered bottles.

Time hours	-100 tailing		-150 tailing		-200 tailing		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
8	0.68	0.38	0.40	
16	0.41	0.20	0.34	0.19	0.33	
24	0.45	0.34	0.40	0.29	0.37	0.28	
32	0.45	0.21	0.39	0.17	0.33	0.13	
40	0.48	0.21	0.40	0.20	0.35	0.15	
48	0.41	0.18	0.33	0.16	0.29	0.12	
48	0.28	Agitation by means of an open bottle revolving horizontally. 1 lb. sodium peroxide per ton of ore used.
48	0.41	
72	0.23	
96	0.21	

The table shows:—

1. The ore can be readily cyanided to 0.68 oz. gold per ton. This gives a recovery of 87.9 per cent.

2. Fine grinding increases the extraction.

3. Longer agitation increases the extraction.

4. The ore ground to -200 mesh and cyanided for 96 hours gives a tailing of 0.21 oz. gold per ton. This means a recovery of 96.3 per cent. This is a very fair recovery but the tailings are too high to be thrown away.

5. Agitation by means of the open bottle gives the same results as agitation by means of the stoppered bottle.

6. Sodium peroxide does not seem to improve the extraction.

Floating and cyaniding.—A number of tests were made using flotation as an aid to cyanidation. The flotation was done in a small Ruth laboratory flotation machine. The cyaniding procedure was similar to that used in the straight cyaniding tests.

Procedure	Concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Floating at -200 mesh with coal oil and pine oil and then cyaniding the flotation tailing for 48 hours.....	20.50	2.22	tr.	tr.
Floating at -200 mesh with TT mixture and then cyaniding flotation tailing for 48 hours.....	83.68	3.91	0.16	0.12
Floating at -200 mesh with heavy hardwood creosote, coal tar and coal tar creosote, and then cyaniding flotation tailing for 48 hours.....	63.72	4.68	0.18	0.08
Cyaniding at -200 mesh for 48 hours and then floating the cyanide tailing with heavy hardwood creosote, coal tar and coal tar creosote.....	2.43	1.03	0.20	0.15
Cyaniding at -200 mesh for 48 hours and then floating the cyanide tailing with TT mixture.....	1.30	1.11	0.24	0.20

The above table shows:—

1. There is difficulty in either getting a representative sample of the ore, or in assaying, as it is hardly possible that the tailing in the first test would run only a trace in gold when two other tests using reagents usually found better for similar ores give tailings of 0.16 and 0.18 oz. gold per ton.

2. Floating either before or after cyanidation improves the extraction a little, but leaves a tailing still too high in gold.

Cyaniding and tabling.—A test was made to see what tabling would do as an aid to cyaniding.

Procedure	Concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Cyaniding at -200 mesh for 48 hours and then tabling cyanide tailing.....	1.30	1.11	0.24	0.20

This test shows that tabling improves the extraction somewhat, but does not give a sufficiently low tailing.

SUMMARY OF EXPERIMENTAL TESTS

1. Cyaniding alone, on the ore submitted, gives a good recovery, but leaves a tailing too high to discard.

2. Flotation, used as an aid to cyaniding, improves the extraction and gives a lower tailing, but the tailing is still too high.

3. Tabling, used as an aid to cyaniding, improves the extraction and gives a lower tailing, but the tailing is still too high.

CONCLUSIONS

It is regrettable that the ore submitted did not assay about the same as the ore being milled by the Wright-Hargreaves Mines, Ltd. The sample tested ran 5.62 ozs. gold per ton. This is equivalent to \$116.17 per ton in gold, and is much higher than the ore that is being milled at present.

On lower grade ore it would be easier to secure a lower tailing, and it is possible that on ore similar to that now being milled at the mine straight cyaniding would work very satisfactorily.

The molybdenite present in the ore seems to have no deleterious effect on the cyanide process.

MICROSCOPIC EXAMINATION OF SELECTED SPECIMENS OF WRIGHT-HARGREAVES ORE.

E. A. Thompson

Two specimens of Wright-Hargreaves ore, selected from a shipment of 100 pounds of high grade, one containing a considerable amount of amorphous molybdenite and graphite, the other practically free from these minerals, were found to be:—

Specimen No. 1.—Telluride ore, a brecciated red syenite porphyry, the fracture faces of which are generally slickensided. Native gold and

gold telluride are visible, closely associated with small amounts of chalcopryrite, amorphous molybdenite and graphite. Pyrite is also present but seems to be more closely associated with the porphyry than the other metallics.

Specimen No. 2 —A brecciated syenite, the fissures of which are filled with carbonate and quartz stringers carrying native gold. Pyrite is quite abundant but does not occur with the fissure material, evidently being of an earlier generation.

Polished Specimens

Specimen No. 1.—The microscopic examination of polished samples shows the ore to have been much fractured. Native gold, gold telluride, small quantities of chalcopryrite, and amorphous molybdenite are found closely associated along the minute fissures. These minerals were deposited at the time of fracturing or at a later period. Invariably wherever telluride is found, native gold is intermixed. Pyrite is present, associated with the gangue, and does not as a rule occur with the other metallics in the fractures, so does not appear to be genetically related to these minerals. A few specks which could not be identified as galena are considered to be lead telluride, altaite.

Specimen No. 2.—Ore has been fractured, the fissures are filled with carbonate and quartz, with native gold associated. Pyrite is also present but is associated with the original rock and is not found along the fissures.

Thin Sections

Specimen No. 1.—Thin sections show the sample to be mostly alkali feldspar, quite weathered. Quartz occurs very sparingly. The rock has been much fractured or brecciated, causing minute fissures, which are invariably filled with amorphous molybdenite, with which native gold and gold telluride are intimately associated. Small quantities of chalcopryrite are also present. The iron pyrite is more closely connected with the feldspar and quartz and seems to have crystallized at the same time. Considerable sericite is associated with the feldspar generally along the contact of the weathered crystals. There is also some chlorite. Graphite is apparently intermixed with the molybdenite. The genetic relationship seems to be a feldspar quartz rock carrying pyrite and then a fracturing and deposition of native gold, telluride, chalcopryrite and molybdenite with a small amount of quartz.

Specimen No. 2.—This ore is mostly alkali feldspar, very much weathered, with considerable sericite, which is generally found around and along the edges of the feldspar crystals. A little chlorite is also present. Quartz of the first generation occurs in minor quantities. The pyrite is closely associated with the quartz and feldspar, and apparently crystallized at the same time as the quartz. The rock has been much fractured. Along the minute fractures or fissures, native gold has been deposited with stringers of carbonate, mostly calcite. In some cases native gold is deposited along the cleavage planes of the calcite. A small quantity of secondary quartz is also found in the fissures.

PLATE 11

Fig. A. No. 1. A typical specimen of the mineral, showing the characteristic color and luster.

Fig. B. No. 2. A typical specimen of the mineral, showing the characteristic color and luster.

Fig. C and D. No. 3. A typical specimen of the mineral, showing the characteristic color and luster.

Fig. E and F. No. 4. A typical specimen of the mineral, showing the characteristic color and luster.

PLATE II

- Fig. A No. 1 ore, polished specimen. Minute fissures carrying native gold, Au, and gold telluride, AuTe_2 , and pyrite, Py, $\times 150$.
- Fig. B No. 1 ore, polished specimen. Fissures carrying amorphous molybdenite, Mo, and graphite, native gold, Au, gold telluride, AuTe_2 , and chalcopyrite, Cp, also showing pyrite, Py, $\times 150$.
- Figs. C and D No. 1 ore, thin sections, ordinary light, showing minute fissures carrying native gold, Au, gold telluride, AuTe_2 , amorphous molybdenite, Mo, graphite and chalcopyrite, Cp, also showing pyrite, Py. The rest of section is weathered alkali feldspar, considerable sericite and a little chlorite.
- Figs. E and F No. 1 ore, thin sections, crossed nicols, showing fractures carrying native gold, Au, amorphous molybdenite, Mo, chalcopyrite, Cp, and pyrite, Py. The rest of the sections show weathered alkali feldspar, f, and abundance of sericite, Ser, and some chlorite.

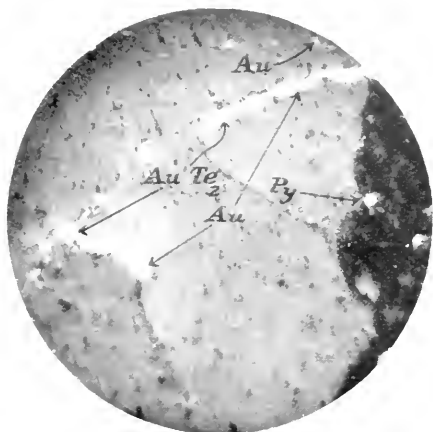


Fig. A.

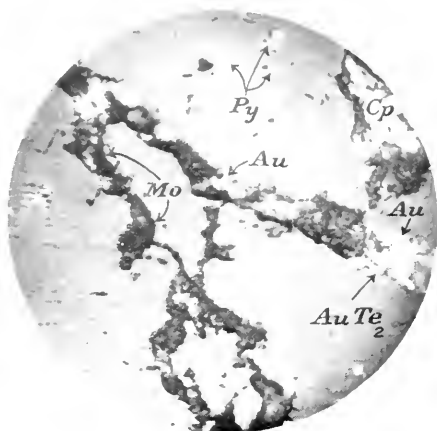


Fig. B.

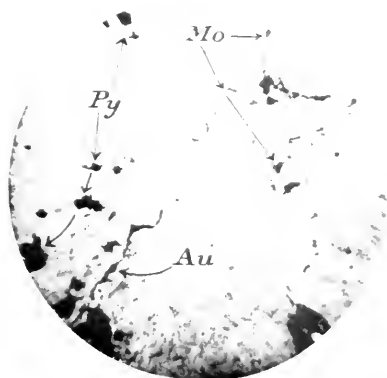


Fig. C.

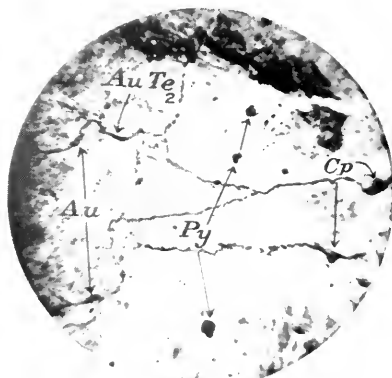


Fig. D.

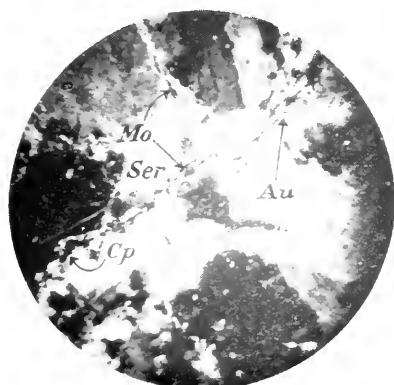


Fig. E.

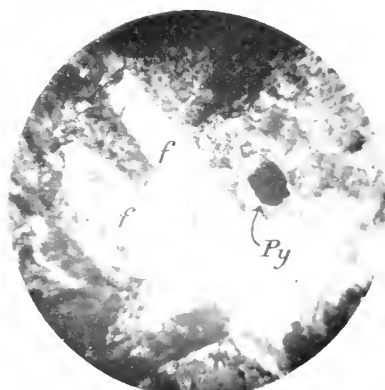


Fig. F.



Fig. 1, 2

The first of the two figures (Fig. 1) shows a cross-section of a cell. The outer ring is dark, and the interior is lighter and textured. The second figure (Fig. 2) shows a similar cross-section, but with a different internal texture.

The third figure (Fig. 3) shows a cross-section of a cell. The outer ring is dark, and the interior is lighter and textured. The fourth figure (Fig. 4) shows a similar cross-section, but with a different internal texture.

The fifth figure (Fig. 5) shows a cross-section of a cell. The outer ring is dark, and the interior is lighter and textured. The sixth figure (Fig. 6) shows a similar cross-section, but with a different internal texture.

The seventh figure (Fig. 7) shows a cross-section of a cell. The outer ring is dark, and the interior is lighter and textured. The eighth figure (Fig. 8) shows a similar cross-section, but with a different internal texture.



PLATE III

- Fig. A No. 2 ore, polished specimen, showing pyrite, Py, and native gold, Au, in a much fractured gangue, x 150.
- Fig. B No. 2 ore, thin section, crossed nicols showing carbonate filling fissures, associated with native gold, Au, balance of section is weathered alkali, feldspar, sericite, a little chlorite and a few grains of pyrite, Py.
- Fig. C No. 2 ore, thin section, ordinary light, showing pyrite, Py, in quartz, Q, and gold, Au, associated with calcite, Ca, balance of section is weathered feldspar and sericite.
- Fig. D No. 2 ore, thin section, crossed nicols showing pyrite, Py, in quartz, Q, gold, Au, associated with calcite, Ca, balance of section is weathered alkali, feldspar and sericite.

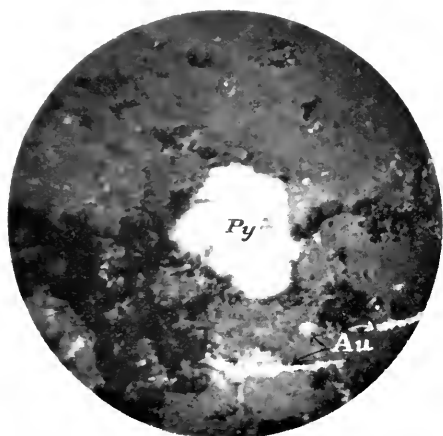


Fig. A.

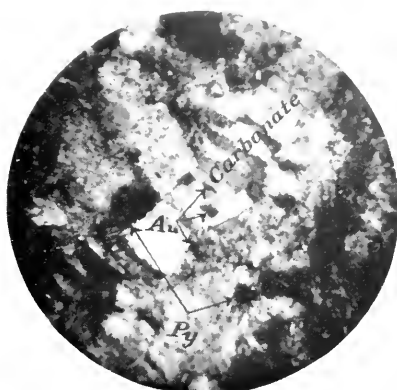


Fig. B.

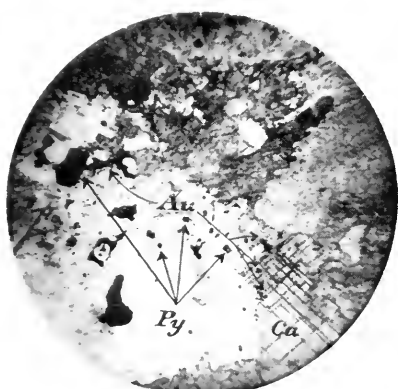


Fig. C.

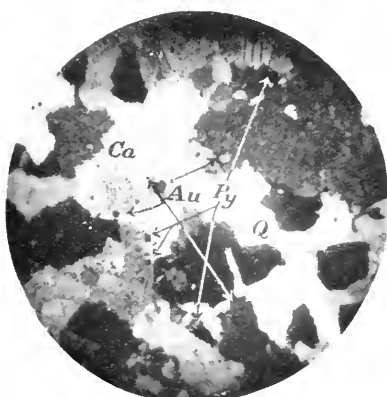


Fig. D.



Analysis of Specimen No. 1

	Per cent
Silica SiO_2	51.30
Alumina Al_2O_3	17.46
Ferric oxide Fe_2O_3	4.64
Lime CaO	1.30
Magnesia MgO	1.80
Soda Na_2O	0.14
Potash K_2O	8.18
Lead oxide PbO	0.14
Copper oxide CuO	1.33
Molybdenum trioxide MoO_3	1.96
Tellurium dioxide TeO_2	0.28=0.23 per cent Te
Gold Au.....	0.33=96.46 oz./ton
Carbon C.....	1.80
Sulphur trioxide SO_3	9.23
Water H_2O	0.50
	100.39

Metallics

	Per cent
Molybdenite MoS_2	2.18
Chalcopyrite CuFeS_2	3.09
Iron pyrite FeS_2	3.29
Altaite PbTe	0.23
Calaverite AuTe_2	0.23
Native gold Au.....	0.23
(Fe_2O_3).....	0.81
Silver Ag.....	not determined

No carbon dioxide, antimony, mercury, bismuth, zinc or barium.

Analysis of Specimen No. 2

	Per cent
Silica SiO_2	55.00
Alumina Al_2O_3	19.89
Ferric oxide Fe_2O_3	5.11
Lime CaO	4.00
Magnesia MgO	1.80
Potash K_2O	7.23
Soda Na_2O	1.78
Sulphur trioxide SO_3	1.68
Carbon dioxide CO_2	1.80
Water H_2O	1.20
Carbon C.....	Nil
	99.49

Carbon dioxide calculated to calcite.....	4.09 per cent
Sulphur trioxide to iron pyrite.....	1.25 "
Calcite.....	4.09 "
Iron pyrite.....	1.25 "
Gold.....	79.02 oz. per ton
Silver, not determined.	

Traces of tellurium, copper, and manganese. No molybdenum or barium.

Report No. 197

GOLD ORE FROM THE PROPERTIES OF THE DASSERAT-ROUYN
GOLDFIELDS, LIMITED, ROUYN TOWNSHIP, TEMISCAMINGUE DISTRICT,
QUEBEC.

R. K. Carnochan

Shipment.—A shipment of 20 bags was received at the Ore Dressing and Metallurgical Laboratories, July 12, 1923, from the Dasserat-Rouyn Goldfields, Limited, 77 Trust Building, Ottawa. The shipment was from the company's claims, 3421a, b, and c, Rouyn township, Témiscamingue district, Quebec.

The shipment was in three lots, and each lot was assayed and tested separately, although the shippers claimed that lot No. 1, consisting of one bag lost in transit and afterwards located and re-shipped, was a part of the original shipment of 19 bags of ore, and that only two lots were submitted, one of ore, and the other consisting of one bag of wall rock.

The lots as received were as follows:—

Lot No. 1 (high grade), 1 bag.....	43.5 pounds net
Lot No. 2 (wall rock), 1 bag.....	54.5 “
Lot No. 3 (low grade), 18 bags.....	856.0 “

Characteristics of the ore samples.—A superficial examination showed the ore of lots 1 and 3 to possess very similar characteristics of mineralization and gangue constituents and it was quite impossible to distinguish which lot was the higher grade ore. The ores of both lots consisted of vein filling, quartz and a dark grey rock material with a considerable amount of iron pyrites. No copper pyrites or tellurides could be detected.

Purpose of experimental tests.—The object in view in submitting these samples to the testing laboratories was to determine if the ore was amenable to treatment with high recoveries of the precious metal values.

Sampling and analysis.—Each lot was crushed and sampled for assay. A considerable amount of flake gold was obtained from the sample of lot No. 1 when preparing it for assay. No flake was obtained from the sample of lot No. 3. The samples showed the following assay values:—

	Gold oz. per ton	Silver oz. per ton
Lot No. 1 (high grade).....	4.75	0.66
Lot No. 2 (wall rock).....	trace	trace
Lot No. 3 (low grade).....	0.31	0.11

Considering lots Nos. 1 and 3 as belonging to the same shipment, this would give an assay value for the ore of—gold 0.525 oz./ton, silver 0.136 oz./ton.

Experimental tests.—Lot No. 2, consisting of wall rock, showed only traces of gold and silver values. No further work was necessary. Cyanide tests were conducted on lot No. 3 followed by tests on lot No. 1.

Test 1, Lot 3.—209·5 grammes of ore, —100 mesh, was agitated for 24 hours in 1000 c.c. water to which had been added 2 grammes of sodium cyanide and 2 grammes soda ash. After agitation the ore was dewatered and washed by means of a suction filter. The results of this test were as follows:—

	Gold	Silver
Cyanide tailing.....	trace	trace
Recovered by cyanidation.....	100·0 per cent	100·0 per cent
Cyanide consumption..... 3·7 lbs. KCN per ton of ore		
Lime “..... 6·0 lbs. CaO “		

Test 2, Lot 3.—205·1 grammes of ore, —200 mesh, was treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tailing.....	trace	trace
Recovered by cyanidation.....	100·0 per cent	100·0 per cent
Cyanide consumption..... 5·0 lbs. KCN per ton of ore		
Lime “..... 6·1 lbs. CaO “		

Test 3, Lot 1.—215 grammes of ore, —100 mesh, were treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tails.....	0·03 oz. per ton	0·31 oz. per ton
Recovered by cyanidation.....	99·4 per cent	53·0 per cent
Cyanide consumption..... 4·6 lbs. KCN per ton of ore		
Lime “..... 5·7 lbs. CaO “		

Test 4, Lot 1.—244 grammes of ore, —100 mesh, were treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tailing.....	0·05 oz. per ton	0·51 oz. per ton
Recovered by cyanidation.....	98·9 per cent	22·7 per cent
Cyanide consumption..... 3·2 lbs. KCN per ton of ore		
Lime “..... 5·7 lbs. CaO “		

Test 5, Lot 1.—216 grammes of ore, —200 mesh, were treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tailing.....	trace	trace
Recovered by cyanidation.....	100.0 per cent	100.0 per cent
Cyanide consumption.....	3.6 lbs. KCN per ton of ore	
Lime ".....	6.1 lbs. CaO "	

SUMMARY OF RESULTS

Test No.	Lot No.	Heads		Mesh	Cyanide tails		Recovery		Consumption	
		Gold oz. per ton	Silver oz. per ton		Gold oz. per ton	Silver oz. per ton	Gold per cent	Silver per cent	Cyanide lbs. per ton	Lime lbs. per ton
1.....	3	0.31	0.11	—100	tr.	tr.	100.0	100.0	3.7	6.0
2.....	3	0.31	0.11	—200	tr.	tr.	100.0	100.0	5.0	6.1
3.....	1	4.75	0.66	—100	0.03	0.31	99.4	53.0	4.6	5.7
4.....	1	4.75	0.66	—100	0.05	0.51	98.9	22.7	3.2	5.7
5.....	1	4.75	0.66	—200	tr.	tr.	100.0	100.0	3.6	6.1

CONCLUSIONS

The ore was very amenable to treatment by the cyanide process. Grinding to 100 mesh for the lower grade ore, and to 200 mesh for the higher grade, gave very low tailings with very high recoveries of the gold and silver values. The consumption of cyanide was high, due, very probably, to the weathered and oxidized condition of surface ore. This consumption should be materially lessened in treating fresh ore from below the zone of weathering and oxidization.

Report No. 198

CONCENTRATION TESTS ON GRAPHITE FROM THE CANADIAN GRAPHITE CORPORATION, GUENETTE, QUEBEC

C. S. Parsons and R. K. Carnochan

Shipment.—A shipment of six bags, containing 300 pounds of graphite ore, was received on June 22, 1923, from the Canadian Graphite Corporation, Guenette, Que. The head office of the company is at 425 Phillips place, Montreal.

Purpose of experimental work.—The purpose of the test work, on this shipment, was to check up the results being obtained in their milling operations, and to determine the percentages of recoverable graphite, in as coarse a form as possible, with high recoveries of the graphite content. The management was not satisfied with the results obtained in their milling operations, and requested the assistance of the Department in improving their practice. Their greatest concern was with their concentrating department, so that only this part of the treatment is considered in this report.

Synopsis of milling operations.—A milling plant had been erected on the property and was in operation for some time. The method first used for the concentration of the graphite was the Spearman film flotation process. This was later changed to the Callow oil flotation process. A flow sheet of the crushing and concentrating departments of the mill is given in Figure 7. The dried concentrates were sent to the finishing department, which is equipped with polishing rolls and screens.

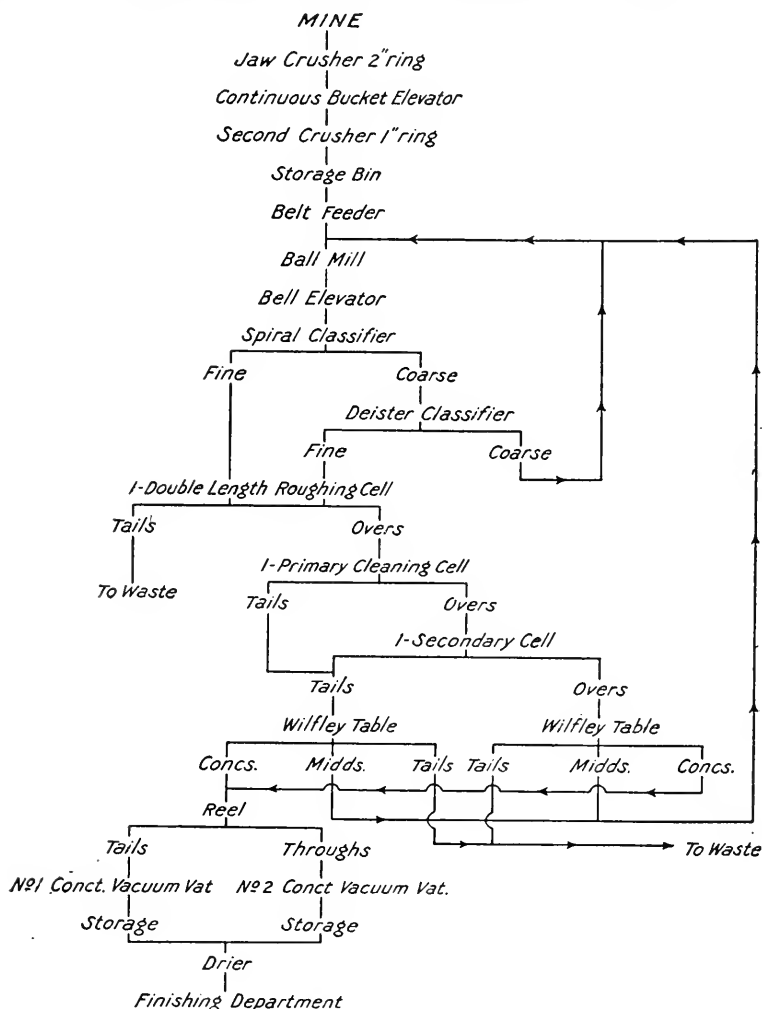


Fig. 7. Flow sheet graphite mill, Canadian Graphite Corporation Limited, Guenette, Que. Crushing and concentrating departments.

Arrangements for experimental work.—Arrangements were made with the company to have their resident manager, Mr. R. E. G. Burroughs, present while the tests were being conducted.

Characteristics of the shipment.—The graphite is of the flake variety, the flakes being of medium size but slightly smaller than those of the Buckingham deposits, the gangue was quite siliceous.

Sampling and analysis.—The entire shipment was sampled by stage crushing and cutting in a riffle sampler. The sample prepared for analysis showed it to contain 16.36 per cent carbon.

EXPERIMENTAL TESTS

Test No. 1

Object of test.—To obtain a high-grade concentrate and a clean tailing.

Procedure.—The flow sheet of Figure 8 was followed in conducting the test.

Reagent used.—To the ball mill while grinding the following reagents were added:—

Kerosene oil.....	1	pound per ton of ore
Pine oil (approx.).....	$\frac{1}{4}$	"

The grinding in the mill was done in a pulp dilution of 1:1. The flotation of the graphite was made in a Ruth laboratory flotation machine.

Results of test.—The results are given in the flow sheet of Figure 8 and in the following table:—

TABLE I
SCREEN ANALYSIS OF FINAL CONCENTRATE

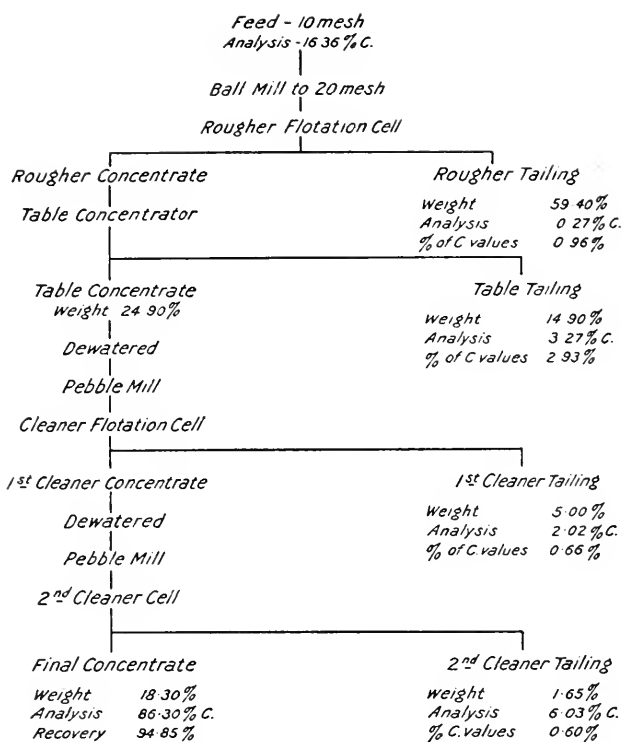
Mesh	Weight		Analysis per cent C.	Distribu- tion of carbon content	Remarks
	Grammes	Per cent			
+100.....	53.2	32.4	93.55	35.5	Recovery of car- bon in final concentrate, 94.85 per cent.
—100+150.....	34.5	21.0	93.72	23.0	
—150.....	76.5	46.6	76.05	41.5	
Total.....	164.2	100.0	84.22	100.0	

Discussion of results.—A high-grade concentrate was obtained, with an exceptionally high recovery, but it is very evident from the screen analysis of the final concentrate that the repeated grinding of the concentrate destroyed an excessive proportion of the large flake in the ore.

Test No. 2

Object of test.—To determine the highest grade concentrate which could be obtained and which would at the same time have the largest proportion of flake remaining on a 100-mesh screen.

Procedure.—The flow sheet of Figure 9 was followed. The flotation concentrate was reground only once and for a much shorter time than in the previous test.



*Note - Table Tailing, 1st and 2nd Cleaner Tailing returned
to Rougher Flotation Cell in practice*

Fig. 8. Flow sheet test No. 1.

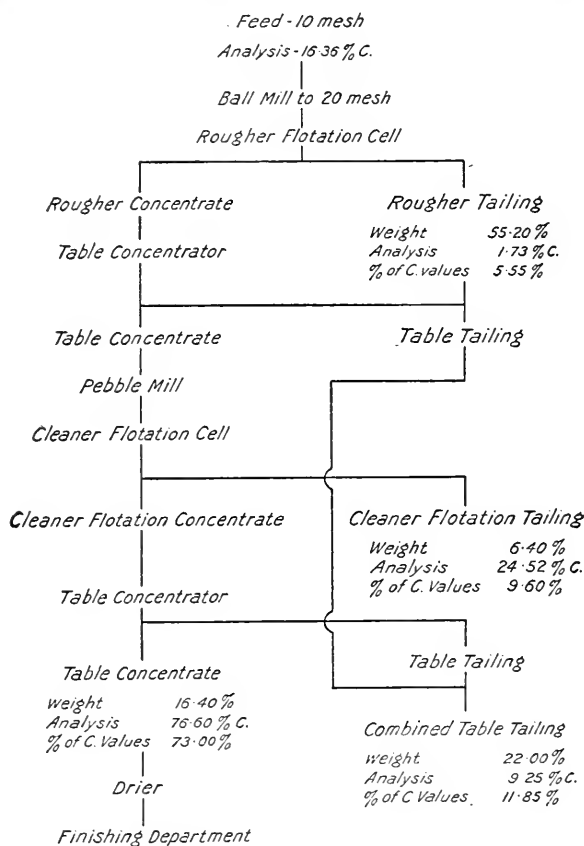
Results.—The results are given in the flow sheet of Figure 9, and in the following table:—

TABLE II
SCREEN ANALYSIS OF FINAL CONCENTRATE

Mesh	Weight		Analysis per cent C.	Distribu- tion of carbon content
	Grammes	Per cent		
+100.....	98.5	63.3	74.05	62.1
+100+150.....	35.0	22.5	79.35	23.5
-150.....	22.3	14.2	76.65	14.4
Total.....	155.8	100.0	100.0

SUMMARY OF TESTS NOS. 1 AND 2

In test No. 2, the final concentrate contained a much larger percentage of the total carbon on a 100-mesh screen than in test No. 1. The figures are 61 per cent of the total in test No. 2 as compared to only 36 per cent in test No. 1. The grade of these two concentrates is, however, the reverse. The material on the 100-mesh screen in test No. 1 assayed 93.55 per cent C. as compared to 74.05 per cent C. in test No. 2. This means that although a lower grade concentrate on the 100-mesh screen was made in test No. 2, almost double the amount of graphitic carbon was obtained.



Note :- Table Tailings and Cleaner Flotation Tailings returned to Rougher Flotation Cell in practice

Fig. 9. Flow sheet test No. 2.

The 93.55 per cent carbon concentrate from test No. 1 still contains grit and must be refined before it can be marketed for lubricating purposes. As both the concentrates must be further refined, the essential point to be determined is where the concentrating should stop and the refining

begin. The coarse flake on 100 mesh being worth three times as much as the finer flake, the efficiency of the milling will depend on the production of the largest possible amount of clean coarse flake that the ore will yield, or put another way, the efficiency will depend on the minimum destruction of the coarse flake during the grinding of the ore to free this flake.

It is quite evident that much more efficient work in this regard can be done in the refinery where the material is treated in batch lots and each operation is under control, than in the concentrator where the operation is continuous.

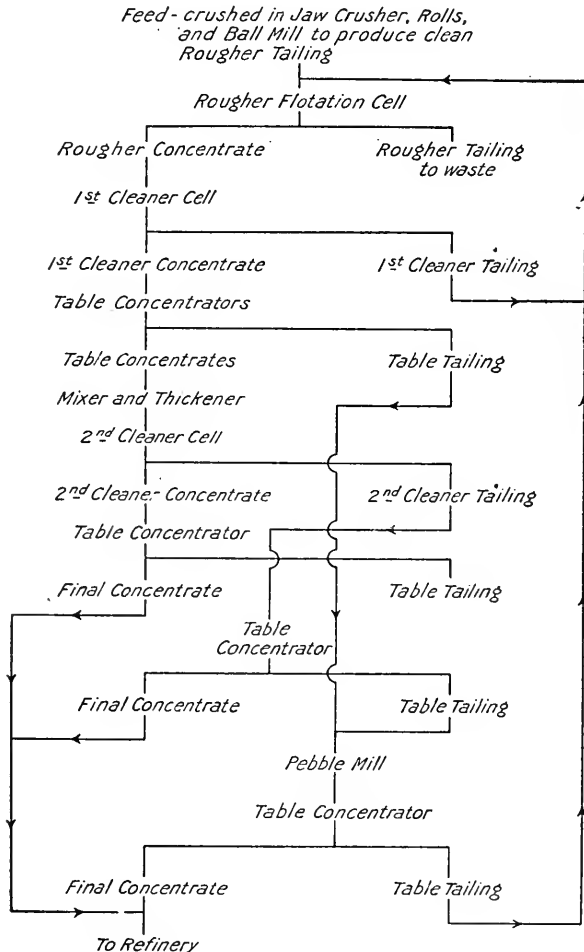


Fig. 10. Flow sheet suggested for treatment of Guenette graphite.

The writers are, therefore, of the opinion that the function of the concentrating department should be to crush the ore only fine enough to obtain a reasonably clean tailing from the flotation rougher cells rather than to produce a concentrate that contains a minimum of free gangue

minerals. Middling products will be produced during the operations, and will consist of true middling, that is, particles of attached gangue and graphite, which the crushing has failed to free, and of particles of gangue which were mechanically entangled in the concentrate. The mill should be designed to divide the middlings into the two classes as far as possible. The class made up of particles in which the gangue is the largest proportion should be returned to the circuit and reground to free the graphite. The second class in which the graphite proportion is the largest, should be made to report in the concentrate, so that the work of freeing the attached pieces of gangue will be thrown on the refinery.

One flow sheet, Figure 10, is given, which it is believed, will produce these conditions, and which is suggested for the concentration of the graphite from the company's property at Guenette, Quebec.

Report No. 199

SELECTIVE FLOTATION OF THE COPPER-NICKEL ORE OF SHEBANDOWAN LAKE DISTRICT, ONTARIO

C. S. Parsons

Shipment.—A shipment of 400 pounds of copper-nickel ore was received January 30, 1923, from Messrs. Jamieson and Peacock, 509 Alworth Building, Duluth, Minn., U.S.A.

Location of deposit.—The Shebandowan nickel deposits are situated at the southwest end of lower Shebandowan lake in the Thunder Bay district, about 73 miles west of Port Arthur, and about $4\frac{1}{2}$ miles south of the Canadian National railway, on mining claims TB 2192, 2204, 2219 and 2240.

Purpose of experimental tests.—Experimental tests were conducted to determine whether the ore could be concentrated and a high-grade copper-nickel concentrate obtained with a satisfactory recovery of the values. Previous work on the ore, the results of which are given in the Summary Report for 1922, page 157, Test No. 172, showed it to contain appreciable amounts of precious metal values, especially platinum and palladium. The tests were conducted to determine also whether the precious metal values were being concentrated with the copper, nickel, and cobalt values, by selective flotation.

Character of the ore.—The ore is mainly iron pyrite, chalcopyrite, and pyrrhotite, in a gangue of alumina, magnesia, and lime silicates. It carries variable percentages of nickel, copper, and cobalt, and precious metals of the platinum group.

Sampling and analysis.—The shipment was crushed to three-quarter inch and cut to 44 pounds. The 44-pound sample was crushed to 20 mesh and a sample of four pounds cut out for analysis. The analysis of this sample was as follows:—

Nickel.....	2.97 per cent	Alumina.....	10.15 per cent
Copper.....	1.65 "	Lime.....	2.50 "
Cobalt.....	0.15 "	Magnesia.....	7.64 "
Iron.....	24.30 "	Gold.....	0.01 oz. per ton
Sulphur.....	15.84 "	Platinum.....	0.03 "
Silica.....	26.55 "	Palladium.....	0.048 "

EXPERIMENTAL TESTS

Tests Nos. 1 and 2 were made to determine whether a high-grade copper-nickel concentrate could be obtained by selective flotation. The procedure was as follows: 1,000 grammes of the ore crushed to 20 mesh was ground in a small ball mill with the flotation reagents for 30 minutes. The pulp density in the mill was one of solids to one of water. The reagents used for test No. 1 were:—

Lime.....	5.0 lbs. per ton
Barretts No. 634.....	0.5 "
TT mixture.....	0.15 "

The reagents used for test No. 2 were:—

Soda ash.....	5.0 lbs. per ton
Barretts No. 634.....	0.5 "
TT mixture.....	0.15 "

The results are given in the following table:—

Test No.	Product	Weight per cent	Analysis			Percentage of values			Total recoveries		
			Cu. per cent	Ni. + Co. per cent	Cu. + Ni. + Co. per cent	Cu.	Ni. + Co.	Cu. + Ni. + Co.	Cu. per cent	Ni. + Co. per cent	Cu. + Ni. + Co. per cent
1	Concentrate.....	25.7	6.35	9.88	16.23	83.0	92.1	88.3	83.0	92.1	88.3
	Tailing.....	74.3	0.45	0.24	17.0	7.9
2	Concentrate.....	30.7	5.00	8.56	15.36	88.3	95.5	92.7	91.1	96.4	94.4
	Middling.....	17.0	0.58	0.30	5.7	1.9			
	Tailing.....	52.3	0.20	0.14	6.0	2.6			

Assuming the nickel mineral to be pentlandite, and the iron pyrrhotite, and taking the analysis of the ore and the tailing of test No. 2, the following percentages are arrived at:—

Mineral	Per cent ore	Per cent tailing
Pentlandite with cobalt.....	14.22	0.64
Chalcopyrite.....	5.71	0.57
Pyrrhotite.....	23.25	11.97
Gangue.....	56.72	86.82

These figures show that 36.1 per cent of the pyrrhotite has been eliminated in the tailing together with the gangue silicates.

CONCLUSIONS FROM EXPERIMENTAL TESTS

A high recovery of the copper, nickel and cobalt values was made in a good grade of concentrate. The recovery of copper is not as good as that obtained on the Sudbury ores, probably due to oxidization. Thirty-six per cent of the pyrrhotite has been eliminated in the tailing, together with the gangue silicates. This elimination of pyrrhotite, if practically barren of precious metal values, together with the gangue minerals, is very desirable for subsequent smelting operations.

Continuous grinding and flotation tests will be made on the ore, and the precious metal values in the flotation products determined, to prove whether these values are recovered in the concentrate.

Report No. 200

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE PROPERTIES OF THE MOLYBDENITE REDUCTION COMPANY, NEAR AMOS, QUEBEC

C. S. Parsons

Shipments.—A shipment of molybdenite ore of about ten tons, in three separate lots, was received at the Ore Dressing and Metallurgical Laboratories, Mines Branch, Department of Mines, Ottawa, August 24, 1923. The shipment consisted of the following lots:—

Lot No.	Number bags	Net weight lbs.	Locality from which samples were taken
1	150	12,064	Shaft dumps from veins Nos. 4 and 4a, La Corne township. Twelve other veins in La Corne township. Two veins in Malartic township.
2	75	6,070	
3	25	2,072	
Total	250	20,186	

The three separate lots were supposed to be average and representative samples of the ore from the company's properties in La Corne and Malartic townships, Abitibi district, Quebec.

Purpose of experimental tests.—The object in conducting experimental tests was to determine whether the ore was amenable to treatment by flotation, with the production of a high-grade marketable product and a high recovery of the molybdenite values. Small scale preliminary tests previously made had shown very encouraging and satisfactory results. Tonnage check tests, or tests on a much larger scale were desired, and if satisfactory results were obtained, the installation of a treatment plant on the company's properties was contemplated.

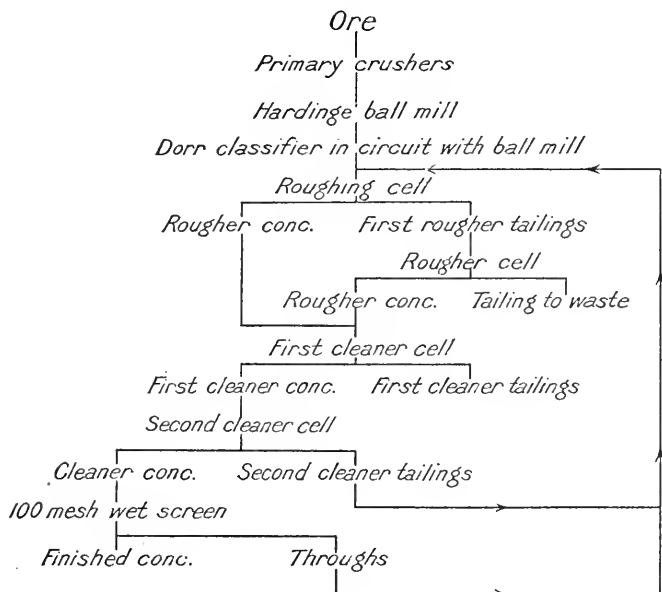
Arrangements for experimental tests.—Arrangements were made whereby the company's consulting engineer, Mr. H. H. Claudet, would co-operate with the officials of the Mines Branch in supervising the whole operations with respect to the experimental work on this ore, and who would submit to the company a separate report describing in detail the operations in conducting the mill runs and the results obtained. A copy of his report was to be filed with the Department for reference.

Characteristics of the ore.—A superficial examination shows the three lots of ore to possess very similar characteristics with respect to molybdenite content and gangue. The latter consists of quartz, a considerable amount of sericite (a thin flake mica), a little iron pyrites and tourmaline. Feldspar and fluorite are also present in small amounts. The molybdenite is associated mainly with the sericite, although it occurs with the quartz and other gangue minerals in smaller amounts. The molybdenite flake is quite heavy but not usually large. Molybdite—the yellow oxide of molybdenum and iron—occurs in such small amounts that it is of no commercial importance.

Sampling and analysis.—The three lots were sampled separately. They were reduced first in a Blake crusher set at three-quarter inch and in rolls set at three-sixteenth inch. A tenth part was cut out by a Vezin sampler. By stage grinding and cutting after each stage of reduction, samples of about 100 grammes ground to 150 mesh were submitted for analysis. The analysis of each lot was determined as follows:—

Lot No.	Total molybdenum (Mo) as molybdenite (MoS_2)	Molybdate MoO_3	Molybdenite MoS_2
	Per cent	Per cent	Per cent
1.....	2.33	0.014	2.316
2.....	1.52	0.027	1.493
3.....	1.88	0.023	1.857

Experimental tests.—The method used for the recovery of the molybdenite content in this ore was one that is considered to be the best practice for concentrating molybdenite ores. It consists of crushing the ore in primary crushers, grinding in ball mills in close circuit with classifiers to the required fineness to free the molybdenite, and floating the molybdenite from its gangue by the oil flotation process. The flow sheet used was as follows, with slight modifications for the various runs:—



Note:—It may be advisable to dispense with the 100 mesh wet screen as this ore contains very little impurities, such as iron pyrites and a high grade concentrate should be produced without the addition of the screen.

Fig. 11. Flow sheet of the Molybdenum Reduction Co's. mine, Amos, Que.

Lot No. 2, consisting of 5,942 pounds, was run first, followed by Lot No. 3, of 1,989 pounds. These two lots were considered as one run, as Lot No. 3 was too small to make a separate run. A partial clean-up of the plant was made before running Lot No. 1, so that the concentrates could be kept separate and an idea of the results on the smaller lot obtained in order to make any changes in operation considered necessary before running the larger lot. Lot No. 1 consisting of 11,826 pounds was run last.

It is a very difficult matter in a run of a few tons, to obtain ideal operating conditions at once. The nearest approach to actual mill conditions is the best results obtained over any one period of time during the operations. It is also difficult to obtain a full recovery of mineral, owing to a considerable portion of the pulp remaining as residue in the ball mill, in the elevators, in the classifiers, in the cells, pipes, etc., and losses which would not occur in practice under continual operation are inevitable in a short mill run. In order to keep these losses as low as possible, it was decided to consider all three lots together, and to make only one clean up at the end. This clean up was figured as a residue, and the molybdenite content in it determined.

The presence of considerable sericite, especially in Lot No. 1, presented another problem in the concentration of this ore. Consisting of thin flakes of mica, if recovered in the operations it would mean a valuable by-product worthy of consideration. In running Lot No. 1, an effort was made to produce a mica product from the second rougher cell, and this product is considered in the recapitulation of the results. Sufficient of this product was not produced to have any bearing on the results of the tests, and as the recovery of the molybdenite was of first importance, this problem was left for future consideration.

Results of experimental tests.—The following tables give the details of the results from the test operations:—

Lot No. 1—

Net weight received.....	12,064 lbs.	
Deduct loss during sampling due to amount of sample, dust, floor sweepings, etc.....	238	
Net weight to ball mill.....		11,826 lbs.

Lot No. 2—

Net weight received.....	6,070	
Deduct loss as above.....	128	
Net weight to ball mill.....		5,942

Lot No. 3—

Net weight received.....	2,052	
Deduct loss as above.....	63	
Net weight to ball mill.....		1,989
Total weight to ball mill.....		19,757

WEIGHTS, ASSAYS AND CONTENT OF ORE TREATED

Lot No.	Net weight to mill	*Assay MoS ₂	Molybdenite content	
			Lbs. per ton	Total content
1.....	Lbs. 11,826	per cent 2.316	46.32	273.89
2.....	5,942	1.493	29.86	88.71
3.....	1,989	1.857	37.14	36.93
Totals and averages.....	19,757	2.02	40.40	399.52

*NOTE.—Molybdenum present as oxide deducted.

WEIGHTS, ASSAYS AND CONTENT OF MOLYBDENITE CONCENTRATES OBTAINED

Product	Run No.	Dry weight	Assay MoS ₂	Content MoS ₂
Concentrates produced from lots Nos. 2 and 3.	1	Lbs. 33.0	Per cent 84.40	Lbs. 27.85
	2	32.0	89.48	28.63
	3	34.0	91.35	31.06
		99.0	88.40	87.54
	4	275.5	91.80	236.39
Concentrates produced from lot No. 1.....				
Total concentrates produced.....		356.5	90.80	323.93

WEIGHTS, ASSAYS, CONTENT AND PERCENTAGES—TOTAL PRODUCTS OBTAINED

Product	Net weight	Assay MoS ₂	Content MoS ₂	Per cent of total MoS ₂ in ore
Total concentrates obtained.....	Lbs. 356.5	Per cent 90.80	Lbs. 323.93	81.08
Mica product from rougher cell lot No. 1.....	164.0	2.37	3.89	0.97
Residue—clean-up of mill.....	1,089.5	3.75	40.82	10.21
Tailings, weight by difference.....	18,147.0	0.10	18.15	4.54
Loss, unaccounted for.....			12.74	3.20
Totals and averages.....	19,757.0	2.02	399.53	100.00

SUMMARY OF RESULTS

The results of the mill run show the following:—

With an average mill feed of 2.02 per cent molybdenite, a concentrate averaging 90.8 per cent molybdenite was obtained with an average tailing over the whole operations of 0.10 per cent molybdenite. This is equivalent to a recovery of about 95 per cent of the molybdenite content of the ore.

The grade of the concentrates produced was above the usual market requirements, which call for 85 per cent molybdenite. As the test proceeded the grade gradually increased from 84.4 per cent molybdenite in the first run, to 91.8 per cent in the fourth run, and there were no doubt

intervals in the latter run where a higher grade was being produced, showing that with continuous operation under more ideal milling conditions, improvement on the grade produced in this test run could be expected.

Considering the grade of the ore, 2.02 per cent molybdenite, the average tailing of 0.10 molybdenite obtained was most satisfactory. In the first run, the tailing was higher than the average, being 0.14 per cent molybdenite, while as the test proceeded a lower tailing was being produced, and in the fourth run a tailing of 0.07 per cent molybdenite was being made, showing that with continuous operation a lower tailing assay could be expected.

The tables, giving the results from the mill run, show an unaccounted loss. This is due to unavoidable spills and leakages during operations on such a small quantity of ore. They also show a residue from clean-up operations, containing considerable molybdenite, that is not recovered in a test of this nature, but which would not occur in the operation of a molybdenite mill. In continuous operations there would be only two products, namely a marketable concentrate, and a tailing, so that the only way to arrive at the percentage recovery is to calculate it from the tailing assay. Taking 0.10 per cent molybdenite, as the average tailing obtained, this would give a recovery of 95.2 per cent of the molybdenite values in the ore.

CONCLUSIONS

The ore received for the mill run is of an exceptionally good grade, slightly higher than the average for molybdenite ores in Canada. It is of a different character from the majority of Canadian ores, in that it is associated mainly with sericite and quartz, and is quite free from other sulphides. In this respect it can be classed as a clean ore.

It is very amenable to concentration by flotation. Its physical character, and the absence of appreciable quantities of other sulphides such as copper and iron, make it an attractive milling ore. It is not difficult to grind, the molybdenite being freed at about 40 mesh. These characteristics permit of an exceptionally high-grade product being produced with a very high recovery of the molybdenite.

Report No. 201

THE CONCENTRATION AND TREATMENT OF ARSENICAL GOLD ORE FROM BATHURST, NEW BRUNSWICK

J. S. Godard

Shipment.—A shipment of 200 pounds of arsenical gold ore was received May 30, 1923, from Dr. L. D. Densmore, Bathurst, N.B. The ore occurs ten miles from Bathurst, on Stevens brook, in Beresford parish, Gloucester co., N.B.

Purpose of experimental tests.—The object, in conducting experimental tests, was to determine whether a marketable arsenopyrite concentrate could be obtained, and also whether the precious metal values could be recovered from the ore or concentrates before shipment to the smelter, or before subsequent refining operations for the recovery of the arsenic. As a general rule smelters and refiners of arsenical ores make no allowance for gold content under \$10 per ton.

Characteristics of the ore.—The ore consists of arsenical pyrites with a considerable amount of iron pyrite in a siliceous gangue. Small amounts of gold and silver values are present.

Sampling and analysis.—The shipment was crushed to one-half inch in a laboratory jaw crusher and a sample of 23 pounds cut out in a Jones riffle sampler. The sample was reduced to 65 mesh and a small sample of 200 grammes cut out. This latter sample was ground to 150 mesh and assayed. Analysis showed the ore to contain:—

Arsenic.....	15.19 per cent
Gold.....	0.07 oz. per ton
Silver.....	1.75 “

EXPERIMENTAL TESTS

Test No. 1.—Cyanidation of the crude and roasted ore

Purpose.—To determine the amount of gold and silver values that could be recovered by cyaniding, first, the crude ore, and, second, the roasted ore.

First: Cyanidation of the crude ore

Procedure.—A sample of 543 grammes was cut from the 23 pounds of the ore crushed to 65 mesh. 100 grammes of this was taken and agitated for 24 hours with the following:—

500 c.c. of 0.25 per cent KCN solution
1.2 lb. per ton of lime

Results.—The solution at the end of agitation was tested for cyanide and lime. It showed a slight acid reaction and complete consumption of the cyanide. Assay of cyanide tailing:—

Gold.....	0.02 oz. per ton
Silver.....	1.05 “

The recoveries were:—

Gold.....	71.4 per cent
Silver.....	40.0 “

Second: Cyanidation of the roasted ore

Procedure.—A sample of 70 grammes of the roasted ore, containing arsenic 1.02 per cent, gold 0.10 oz. per ton, silver 2.46 oz. per ton, was agitated for 24 hours with the following:—

500 c.c. of 0.25 per cent KCN solution
1 lb. per ton of lime

Results.—On completion of the test the solution was found to be acidic and showed a complete consumption of the cyanide. Assay of cyanide tailing:—

Gold.....	0.04 oz. per ton
Silver.....	1.86 “

Recoveries were:—

Gold.....	60.0 per cent
Silver.....	24.4 “

Summary.—In both cases the recoveries are low. The solutions were found to be acidic at the completion of the tests. The acidity would destroy the free cyanide which accounts for the poor recoveries. Lower recoveries on the roasted ore are due to the more rapid consumption of the free cyanide by soluble sulphates formed during the roasting.

Test No. 2 — Table concentration

Purpose.—To determine whether the ore could be concentrated by tabling with the production of a high-grade arsenical product, carrying the gold and silver values.

Procedure.—1600 grammes of the ore previously crushed to one-half inch was further ground to 40 mesh and sized on 80 mesh, making two sizes, viz.: 720 grammes $-40+80$ and 880 grammes -80 mesh. Each size was tabled separately.

Results.—Are shown in following table:—

Product	Weight per cent	Analysis			Percentage of values		
		As. per cent	Au. oz.	Ag. oz.	As.	Au.	Ag.
Concentrate $+80$	27.8	24.00	0.10	1.70	43.5	41.5	28.0
Concentrate -80	26.7	24.71	0.09	1.83	43.8	35.9	29.0
Tailing $+80$	15.4	2.13	0.01	0.60	2.1	2.3	5.5
Tailing -80	18.8	3.94	0.03	1.52	4.8	8.4	16.9
Slimes.....	11.3	8.02	0.07	3.07	5.9	11.8	20.6

Ratio of concentration..... 1 : 1.8

Average grade of concentrate...As. 24.35 per cent Au. 0.10 oz. per ton Ag. 1.76 oz. per ton

Recoveries in concentrate.....As. 87.1 “ Au. 76.5 per cent Ag. 57.0 per cent

Summary.—Table concentration does not give a very high-grade concentrate, due to the presence of a considerable amount of iron pyrite which reports with the arsenopyrite. The gold content is slightly higher in the coarse concentrate, but the silver tends to go with the finer material. The slimes contain 20.6 per cent of the silver values.

Test No. 3.—Jig and table concentration

Purpose.—To determine the grade of the concentrate and recovery of the values by jig and table concentration of the sized products.

Procedure.—A sample of 40 pounds of the half-inch material was cut out and reduced to 4 mesh. It was closely sized, using Tyler standard screens, from 4 mesh to 100 mesh. Sizes coarser than 20 mesh were jigged, sizes finer than 20 mesh were tabled. Jig tailings were re-ground to 20 mesh, sized and tabled. Concentrates and tailings were made from each size.

Results.—The final results obtained on the combined jig and table concentrates and tailings are shown in the following table:—

Product	Weight per cent	Analysis			Percentage of values		
		As. per cent	Au. oz.	Ag. oz.	As.	Au.	Ag.
Combined concentrate.....	63.8	22.65	0.083	2.06	91.1	87.3	86.7
Combined tailings.....	31.9	2.40	0.024	0.62	8.9	12.7	13.3

Ratio of concentration..... 1:1.56

Summary.—Compared with test No. 2, jig and table concentration of the sized material gives a higher recovery of the arsenic, gold, and silver values, but the grade of the concentrate so produced is not as high as by straight table concentration.

Test No. 4—Cyanidation of the coarse concentrate

Purpose.—To determine whether the gold and silver values could be recovered by leaching with cyanide solution.

Procedure.—500 grammes of concentrates –28+35 mesh were leached with 0.25 per cent KCN solution. The first leach was for 24 hours. A second leach for 19 hours was given with partly used cyanide solution. The residue was washed twice, first with 0.05 per cent KCN solution and the second wash with 0.04 per cent KCN solution. The residue was then washed free from cyanide, dried and assayed.

Results.—The results of the test are shown below:—

Metal	Before leaching	After leaching	Recoveries
	oz. per ton	oz. per ton	per cent
Gold.....	0.083	0.046	44.5
Silver.....	2.06	1.21	41.3

Cyanide consumption..... 8 lbs. per ton
Lime “..... 9.5 “

Test No. 5—Cyanidation of the roasted concentrate

Purpose.—To determine whether the gold and silver values could be recovered from the roasted concentrate by cyanidation.

Procedure.—A series of preliminary tests were made on the roasted concentrate to determine the quantity of lime to be added to maintain a working protective alkalinity. It was found that lime equivalent to 18 lbs. per ton of roasted concentrate was required. A series of tests were made giving the roasted concentrate water washes to remove soluble sulphates and thus reduce the amount of lime required for cyanidation. Tests were made on the washed roasted concentrate using 0.05 per cent KCN solution and lime equivalent to 8, 10 and 12 lbs. per ton respectively. The time of agitation was 24 hours.

Results.—The concentrate previous to roasting contained: Gold, 0.083 oz. per ton, silver 2.06 oz. per ton. After roasting: Gold, 0.094 oz. per ton, silver 2.60 oz. per ton.

Sample No.	Before cyanidation		After cyanidation		Recoveries	
	Au. oz.	Ag. oz.	Au. oz.	Ag. oz.	Au. per cent	Ag. per cent
1.....	0.094	2.60	0.023	2.08	75.6	20.0
2.....	0.094	2.60	0.023	1.88	75.6	27.7
3.....	0.094	2.60	0.023	2.14	75.6	17.7

The cyanide and lime consumption was as follows:—

Sample No.	KCN			CaO		
	Per cent before	Per cent after	Lbs. per ton consumed	Lbs. per ton before	Per cent after	Lbs. per ton consumed
1.....	0.05	0.020	1.80	8	0.0065	7.63
2.....	0.05	0.020	1.80	10	0.025	8.51
3.....	0.05	0.016	2.04	12	0.024	10.57

Test No. 6—Cyanidation of the fine concentrate

Purpose.—To determine the recoveries of gold and silver values, using varying strengths of solution and lime equivalent to 15 lbs. per ton.

Procedure.—500 grammes concentrates were ground to pass 100-mesh screen. Three lots of 100 grammes each were agitated for 24 hours in 0.05, 0.10 and 0.20 per cent KCN solutions respectively, with lime equivalent to 15 lbs. per ton in each case.

Results.—The results were as follows:—

Sample No.	Before cyanidation		After cyanidation		Recoveries	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. per cent	Ag. per cent
1.....	0.083	2.06	0.047	1.45	43.4	29.6
2.....	0.083	2.06	0.052	1.48	37.3	28.2
3.....	0.083	2.06	0.047	1.05	43.4	49.0

The cyanide and lime consumptions were as follows:—

Sample No.	KCN			CaO		
	Per cent before	Per cent after	Lbs. per ton consumed	Lbs. per ton before	Per cent after	Lbs. per ton consumed
1.....	0.05	Nil	3.0	15	neutral	15.0
2.....	0.10	0.01	5.4	15	0.0023	14.9
3.....	0.20	0.024	10.6	15	0.0024	14.8

Summary of cyanide tests on concentrate.—The best results were obtained by giving the roasted concentrate a water wash to remove soluble sulphates. Cyanidation of the washed concentrate gave a recovery of the gold of 75·6 per cent. Considering the low value of the gold content in the concentrate, it is questionable if it would pay for its recovery.

Flotation and Table Concentration Tests

Purpose.—To determine whether the ore could be concentrated by flotation and tabling with the production of a high-grade arsenical product carrying the gold and silver values.

Procedure:—

- Test No. 9.—1,000 grammes of the ore ground wet in ball mill to 48 mesh with flotation reagents; floated in laboratory Ruth machine making concentrate and tailing. Tailing concentrated on laboratory Wilfley table.
- Test No. 10.—Same procedure as test No. 9.
- Test No. 12.—1,000 grammes of the ore ground wet in a ball mill to 48 mesh with flotation reagents; floated in laboratory Ruth machine making concentrate and tailing; rougher concentrate re-cleaned making concentrate and middling; pulp density 1·3.
- Test No. 13.—1,500 grammes of the ore ground wet in ball mill to 48 mesh and screened on 100-mesh screen; the +100-mesh material was concentrated on laboratory Wilfley table making a clean concentrate and a tailing. The table tailing was mixed with the -100-mesh material and floated in laboratory Ruth machine making a concentrate and tailing. The rougher concentrate was re-cleaned making a concentrate and middling.
- Test No. 14.—1,000 grammes of the ore ground wet in ball mill to 100 mesh with flotation reagents; floated in laboratory Ruth machine, the rougher concentrate being re-floated making a concentrate, middling and tailing.
- Test No. 15.—642 grammes of the ore ground wet in ball mill to 48 mesh with flotation reagents; floated in laboratory Ruth machine, the rougher concentrate being re-floated making a concentrate, middling and tailing.

Reagents Used for Flotation Tests

Test No. 9.—Tar oil.....	0·50 lb. per ton
Fumol No. 6.....	2 drops
H ₂ SO ₄	18 lb. per ton
Test No. 10.—Barretts No. 634.....	0·50 lb. per ton
Fumol No. 6.....	2 drops
H ₂ SO ₄	18 lb. per ton
Test No. 12.—Water-gas tar, 50 per cent; Coal tar creosote, 50 per cent.....	0·50 lb. per ton
Fumol No. 6.....	2 drops
H ₂ SO ₄	15 lb. per ton
Test No. 13.—Barretts No. 634.....	0·50 lb. per ton
Fumol No. 6.....	2 drops
H ₂ SO ₄	18 lb. per ton
Test No. 14.—Coal tar, 40 per cent; Coal tar creosote, 60 per cent	0·50 lb. per ton
Pine oil No. 350.....	2 drops
H ₂ SO ₄	18 lb. per ton
Test No. 15.—Coal tar.....	0·50 lb. per ton
Fumol No. 6.....	2 drops
H ₂ SO ₄	18 lb. per ton

Results.—The results of the tests are shown in the following table:—

Test No.	Product	Weight p.c.	Analysis				Per cent of values		
			As. p.c.	Fe. p.c.	Au. oz.	Ag. oz.	As.	Au.	Ag.
9	Flotation concentrate.....	55.0	30.52	35.75	0.10	1.85	88.0	89.9	63.9
	Table ".....	2.8	27.09	32.93	0.07	3.38	4.0	3.2	5.9
	Tailing.....	42.2	3.63	0.01	1.14	8.0	6.9	30.2
10	Flotation concentrate.....	58.6	30.07	34.74	0.11	1.95	91.4	96.1	73.1
	Table ".....	0.9	29.16	32.93	0.07	2.62	1.3	0.9	1.5
	Tailing.....	40.5	3.53	0.005	0.98	7.3	3.0	25.4
12	Flotation concentrate.....	39.3	32.00	65.0
	" middling.....	22.5	25.40	29.5
	" tailing.....	38.2	2.79	5.5
13	Table concentrate.....	17.3	30.91	35.35	31.8	25.4 Fe.
	Flotation concentrate.....	27.2	26.83	35.55	43.1	40.3 Fe.
	" middling.....	15.9	19.76	30.70	18.7	20.3 Fe.
	" tailing.....	39.6	2.69	8.48	6.4	14.0 Fe.
14	Flotation concentrate.....	47.3	29.22	35.75	76.4	69.6 Fe.
	" middling.....	16.3	14.45	21.21	13.0	14.7 Fe.
	" tailing.....	36.4	5.41	10.20	10.6	15.7 Fe.
15	Flotation concentrate.....	46.8	32.91	75.4
	" middling.....	17.0	18.65	15.5
	" tailing.....	36.2	5.12	9.1

SUMMARY OF FLOTATION AND TABLE TESTS

The results of the above tests show that an arsenical concentrate, containing 30 per cent arsenic, 0.11 oz. gold and 1.95 oz. silver, can be obtained by flotation at 48 mesh and tabling of the flotation tailing. The recoveries of the values in this concentrate are about 92 per cent of the arsenic, 93 per cent of the gold, and 70 per cent of the silver values in the ore. The amount of sulphuric acid necessary to float the arsenopyrite is high, but no satisfactory flotation results were obtained without its use in liberal quantities.

Flotation at 48 mesh gives better results than when finer grinding is done.

GENERAL CONCLUSIONS

It does not seem advisable to cyanide the crude ore, or the concentrates either crude or roasted, as they do not contain sufficient values to pay for the cost of operation. The recoveries are low and the cyanide consumption high. The best results were obtained on the washed roasted concentrate where 76.5 per cent of the gold values were recovered.

The simplest and cheapest method of concentration is flotation at 48 mesh and tabling of the flotation tailing. An arsenical concentrate is produced, containing 30 per cent arsenic with a recovery of 92 per cent of the arsenic values. This concentrate would probably stand shipment to the arsenic smelters at present market prices for arsenic content in crude concentrate.

Report No. 202

THE TAILING DUMPS OF THE NAUGHTON GOLD MINE, NAUGHTON,
ONTARIO

C. S. Parsons

The Naughton gold mine is situated 12 miles from Naughton station on the Soo branch of the Canadian Pacific railway. There is a good wagon road to the mine from the station, but at present a number of the bridges have been washed out, making the road impassible except in winter. The best method of reaching the mine is by wagon road from Sudbury or Naughton station to the Indian reserve on Black lake, and by canoe across Black lake, Round lake and Long lake, on which route there are two short portages. The mine is about a mile southeast of Long lake.

A visit was made to the mine on August 3, 1923, for the purpose of sampling the tailing dumps and estimating the available tonnage that could be reclaimed. It was reported that the dumps contained a large tonnage, assaying 10 per cent arsenic and 0.2 oz. per ton in gold.

Observations.—The mine had been abandoned, the mill and other buildings torn down and the equipment removed. The tailings were in two main dumps, in a creek bed below the mill. They were held back by a retaining wall 100 feet below the mill, and by a dam one-half mile farther down the creek towards Long lake, and into which the creek entered. A certain amount of concentration had taken place in the dumps, the coarser sulphide minerals remaining in the upper dump, the sand, gangue minerals and slime being washed down by the creek to the lower dump.

Estimate of recoverable tonnage.—The upper dump covers an area of 13,800 square yards and in places would be 12 to 15 feet deep. Figuring on an average depth of 7 feet, this dump would contain approximately 40,000 tons that could be reclaimed. The lower dump covers a considerable area along the banks of the creek, and in the basin above the dam. It is estimated that the area is about 35,000 square yards, and figuring on an average depth of 3.5 feet, the dump would contain approximately 55,000 tons that could be reclaimed. The total reclaimable tonnage from the two dumps would be approximately 95,000 tons.

Samples and analysis.—Eight samples, numbers 1 to 8 inclusive, were taken from the upper dump, and four samples, numbers 9 to 12 inclusive, from the lower dump. These samples were so taken that they should be fairly representative. The analyses are as follows:—

TABLE I

Sample No.	Arsenic per cent	Iron per cent	Sulphur per cent	Gold oz. per ton	Silver oz. per ton
1.....	2.09	7.93	6.22	0.06	0.02
2.....	3.23	11.59	9.23	0.12	0.02
3.....	7.81	14.64	11.38	0.17	0.04
4.....	7.49	20.74	19.58	1.14	0.12
5.....	7.53	13.83	9.75	0.26	0.04
6).....	3.02	9.96	7.51	0.08	0.02
7).....					
8.....	1.53	7.32	5.94	0.06	trace
9.....	1.06	4.88	4.64	0.04	trace
10.....	5.22	17.98	15.44	0.18	0.04
11.....	0.50	4.67	3.66	0.04	trace
12.....	1.74	5.69	3.24	0.04	trace

Composite samples for concentration tests.—Composite samples were made up from the samples from the two dumps, for the purpose of making concentration tests to determine the grade of concentrate that could be produced by a simple means of water concentration and to determine the recovery of the arsenic and gold values. These samples showed the following analyses:—

TABLE II

Description of sample	Arsenic per cent	Iron per cent	Sulphur per cent	Gold oz. per ton	Silver oz. per ton
Composite sample upper dump 1-8	3.92	8.90	12.02	0.15	trace
Composite sample lower dump 9-12.....	2.24	8.90	8.94	0.08	trace

Concentration tests.—The composite samples were run on a small Wilfley table without classification or sizing. The results are shown in the following table:—

TABLE III

Product	Weight per cent	Analysis					Per cent of values	
		As. per cent	S. per cent	Fe. per cent	Au. oz. per ton	Ag. oz. per ton	As.	Au.
Table test on combined samples Nos. 1-8 in- clusive—								
Concentrate.....	14.4	15.28	31.27	41.68	0.48	trace	62.9	57.3
Tailings.....	85.6	1.52	4.46	6.51	0.06	trace	37.1	42.7
Table test on combined samples Nos. 9-12 in- clusive—								
Concentrate.....	8.5	6.15	36.0	42.31	0.26	trace	22.7	28.8
Tailings.....	91.5	1.94	3.79	5.30	0.06	trace	77.3	71.2

CONCLUSIONS

The Naughton tailing dumps contain an estimated tonnage that could be reclaimed of 95,000 tons, 40,000 tons of which are in the upper dump, and 55,000 tons in the lower dump.

The upper dump shows an average analysis of about 4 per cent arsenic and 0.15 oz. per ton gold.

The lower dump shows an average analysis of about 2.25 per cent arsenic and 0.08 oz. per ton gold.

A concentration test on a composite sample from the upper dump shows a ratio of concentration of 1 : 7, a grade of concentrate 15.28 per cent arsenic and 0.48 oz. per ton gold, with a recovery of 62.9 per cent of the arsenic values and 57.3 per cent of the gold values.

A concentration test on a composite sample from the lower dump shows a ratio of concentration of 1 : 12, a grade of concentrate 6.15 per cent arsenic and 0.26 oz. per ton gold, with a recovery of 22.7 per cent of the arsenic values and 28.8 per cent of the gold values.

The results of the concentration tests show that a high-grade arsenical concentrate, with fair recoveries, cannot be obtained by simple gravity methods on account of the iron sulphides present in the tailings.

The analyses and results of the concentration tests indicate that it would not pay to reclaim and concentrate the dumps, especially the lower dump, as the grade of the arsenical concentrates produced would be too low to stand the cost of concentration, treatment charges, and transportation to the smelter.

Report No. 203

CONCENTRATION OF THE ZINC-IRON MIDDLING FROM THE DUMP AT NOTRE-DAME DES ANGES STATION, QUEBEC

C. S. Parsons

Shipment.—A shipment of one bag of zinc-iron middlings, weighing 109 pounds, was received at the Ore Testing Laboratories, October 5, 1923, from the British Metals Corporation, 263 St. James St., Montreal.

Location of dump.—The dump from which this sample was taken is situated at Notre-Dame des Anges station, Que., and is estimated to contain about 40,000 tons.

Character of sample.—The material, in this dump, is a zinc-iron middling product, produced by a former concentration mill in which the ore from the Notre Dame mine was treated. The material is a mixture of zinc, iron, and lead sulphides that have been freed to a large extent by the previous milling operations. There are small amounts of gold and silver associated with the sulphides.

Sampling and analysis.—A sample was taken of the shipment, and the report of the analysis is as follows:—

Zinc.....	24.54 per cent
Lead.....	2.51 “
Iron.....	25.02 “
Gold.....	0.10 oz. per ton
Silver.....	4.56 “

Purpose of experimental work.—A product of the above analysis is not marketable. The purpose of the experimental work was to determine whether the zinc, lead, gold, and silver values could be concentrated by the elimination of the iron sulphides and gangue minerals.

Experimental work.—Two methods of solving the problem were tried—flotation, and magnetic separation. The latter method was supplemented by a magnetic roast. One test only was run using magnetic separation, as this method was given an extensive trial in the past and met with very little success. Flotation processes offered the greatest possibility, and it was on these that the work was concentrated.

Magnetic separation.—The ore was roasted at approximately 800° F. in a small rotary kiln. The kiln was fed and discharged continuously. The results of this test are shown in the accompanying tables.

Conclusions from magnetic separation test.—A high recovery of the zinc can be obtained by this method if the roast is properly controlled.

The zinc concentrate, however, will contain all the siliceous material, which prevents a high-grade product from being obtained. It is possible that tabling would raise the grade of the zinc product without much loss.

Flotation.—A concentrate was desired which would contain over 40 per cent zinc. Twenty-five tests were conducted, using various combinations of reagents. The zinc concentrate in a few of the tests was tabled to remove the lead. The results are given in the accompanying tables.

Conclusions from flotation tests.—The results of these tests are not conclusive, they simply indicate that certain recoveries can be obtained under certain conditions. It is possible, and quite probable, that slightly better results may be obtained. The material is, however, difficult to float. The lead is partly oxidized, and there are considerable quantities of soluble salts present. There is no doubt that a mill starting operations on this dump material would have trouble obtaining and maintaining the correct flotation conditions. It would be extremely optimistic to assume that any better results can be obtained in practice than were obtained in tests Nos. 2, 23, 24, and 25. It is the writer's opinion that small scale batch tests on material of this kind are not reliable, and that tonnage check tests should be conducted before a mill is erected.

REAGENTS USED IN CONDUCTING FLOTATION TESTS

Test No. 1.—Ore —60 mesh.....	1000 grammes (approx.)
Lime.....	1 gramme
Soda ash.....	2 grammes
Copper sulphate.....	1 gramme
Barretts No. 634.....	10 drops
TT mixture to cell.....	4 “
Pine oil to cell.....	1 drop
Rougher concentrate recleaned.	
Test No. 2.—Ore —80 mesh.....	1000 grammes (approx.)
Lime.....	0.5 gramme
Soda ash.....	2.0 grammes
Copper sulphate.....	1.0 gramme
Barretts No. 634.....	15 drops
TT mixture to cell.....	6 “
Pine oil to cell.....	1 drop
Rougher concentrate recleaned.	
Test No. 3.—Ore —90 mesh.....	1000 grammes (approx.)
Lime.....	2.0 grammes
Soda ash.....	2.0 “
Copper sulphate.....	1.0 gramme
Barretts No. 634.....	0.5 c.c.
TT mixture to cell.....	6 drops
Pine oil to cell.....	1 drop
Rougher concentrate recleaned.	
Test No. 4.—Ore —90 mesh.....	1000 grammes (approx.)
Soda ash.....	2.5 grammes
Copper sulphate.....	1.0 gramme
Barretts No. 634.....	0.5 c.c.
TT mixture to cell.....	6.0 drops
Pine oil to cell.....	2.0 “
Rougher concentrate recleaned.	

- Test No. 5.—Ore —100 mesh..... 1500 grammes (approx.)
 Lime..... 0.75 gramme
 Soda ash..... 3.0 grammes
 Copper sulphate..... 1.5 gramme
 Barretts No. 634..... 20 drops
 TT mixture to cell..... 6 “
 Pine oil to cell..... 1 drop
 Rougher concentrate recleaned.
- Test No. 10.—1000 grammes ore, 10 minutes in ball mill.
 Lime..... 0.8 gramme
 Soda ash..... 2.5 grammes
 Copper sulphate..... 1.0 gramme
 Barretts No. 634..... 15 drops
 TT mixture..... 6 “
- Test No. 11.—1000 grammes ore, crushed for 15 minutes.
 Sodium sulphide..... 5 lbs. per ton
 Sodium sulphite..... 2 “
 Barretts water-gas tar..... 6 drops=0.35 lb. per ton
 YZ mixture..... 0.25 lb. per ton
 YZ added in flotation machine, also sodium sulphide.
 It was allowed to mix for 2 minutes before floating.
 NOTE.—Necessary to add 0.36 lb. more water-gas tar.
 Very hard to float anything.
- Test No. 13.—1000 grammes ore, 15 minutes in ball mill.
 Barretts water-gas tar..... 0.5 lb. per ton
 50 per cent acid..... 5 c.c.
 Pine oil..... 2 drops
 XY mixture..... 6 “
- Test No. 12.—1000 grammes ore, 15 minutes in ball mill.
 No. 34 gravity fuel oil and
 Salt cake added in Ruth machine.
 Concentrate high in iron and tails high in zinc.
- Test No. 14.—1000 grammes ore, 15 minutes in ball mill, —80 mesh.
 Barretts water-gas tar to ball mill..... 0.5 drop
 XY to cells..... 6 drops
 Pine oil to cells..... 2 “
 Floated rougher concentrate. In cleaning rougher concentrate very little floated. Added 3 drops more XY, very little additional float obtained.
- Test No. 15.—1000 grammes, 15 minutes in ball mill.
 Barretts water-gas tar to ball mill..... $\frac{1}{2}$ lb. per ton
 Na₂S to cell..... 7 “
 YZ to cells..... 4 drops
 Pine oil to cells..... 2 “
 Coal tar creosote..... 6 “
 NOTE.—Iron floated, apparently zinc in tails. No mid-dling made.
- Test No. 16.—1000 grammes ore, 15 minutes in ball mill.
 CuSO₄ in ball mill..... 4 lb. per ton
 Water-gas tar..... 18 drops
 YZ to cells..... 6 “
 Pine oil..... 2 “
 Heavy froth obtained. Added 1 drop pine oil in recleaning rougher concentrate.

Test No. 17.—1000 grammes, 15 minutes in ball mill.

CuSO ₄ in ball mill.....	4 lb. per ton
Water-gas tar.....	9 drops
XY in cells.....	6 “
Pine oil in cells.....	2 “

Test No. 18.—1000 grammes, 15 minutes in ball mill.

YZ mixture to ball mill.....	0.15 c.c.
Soda ash to ball mill.....	2.5 grammes
Copper sulphate.....	2.0 “
YZ to cleaner cell.....	1 drop

NOTE.—This test was run to determine effect of presence of soda ash in pulp compared to last two tests Nos. 16 and 17 when no soda ash was used.

Test No. 19.—1000 grammes, 20 minutes in ball mill.

Barretts water-gas tar to mill.....	6 drops
CuSO ₄ to ball mill.....	2 grammes
Soda ash.....	2.5 grammes
YZ mixture to cells.....	4 drops
Pine oil to cells.....	1 drop
Made clean looking concentrate and tailing.	

Test No. 20.—1000 grammes, 30 minutes in ball mill.

Barretts water-gas tar to ball mill.....	6 drops
CuSO ₄ to ball mill.....	2 grammes
Soda ash to ball mill.....	2.5 grammes
XY mixture to cells.....	4 drops
Pine oil.....	1 drop
For comparison of YZ and XY mixtures.	

Test No. 21.—1000 grammes ore, 20 minutes in ball mill.

Barretts water-gas tar to mill.....	6 drops
Soda ash to mill.....	2.5 grammes
YZ mixture to cells.....	4 drops
Pine oil to cells.....	1 drop

NOTE.—This test was run to determine effect of CuSO₄.

Test No. 22.—1000 grammes ore, 30 minutes in ball mill.

Barretts No. 634 to ball mill.....	6 drops
CuSO ₄ to ball mill.....	2 grammes
Soda ash.....	2.5 grammes
YZ to cells.....	4 drops
Aldol to cells.....	1 drop

NOTE.—This test was run to determine effect of fine grinding. The pulp was not alkaline.

Test No. 23.—1000 grammes ore, 25 minutes in ball mill, —100 mesh.

Barretts No. 634 to ball mill.....	6 drops
CuSO ₄ to ball mill.....	2 grammes
Lime.....	0.5 gramme
Soda ash.....	2.5 grammes
YZ to cells.....	4 drops
Aldol to cells.....	1 drop

NOTE.—Pulp was not alkaline, so added 1 gramme more soda ash to cell. Pulp turned alkaline. This looks the best test of all, probably due to alkalinity with soda ash. Other tests were not fully alkaline.

Test No. 24.—1000 grammes ore, 25 minutes in ball mill.

Water-gas tar to ball mill.....	3 drops
Soda ash to ball mill.....	3.5 grammes
CuSO ₄ to ball mill.....	2.0 "
Lime to ball mill.....	0.5 gramme
YZ to cells.....	3 drops
Soda ash to cells.....	0.5 gramme
Aldol to cells.....	1 drop

NOTE.—Pulp was not alkaline, so added 0.5 gramme soda ash to make it so.

Test No. 25.—1000 grammes ore, 25 minutes in ball mill.

Water-gas tar to ball mill.....	3 drops
Soda ash to ball mill.....	4.5 grammes
CuSO ₄ to ball mill.....	2.0 "
Lime to ball mill.....	0.5 gramme
YZ to cells.....	2 drops

Aldol added to cleaner cell only.

NOTE.—Pulp was alkaline from ball mill.

Results of Magnetic and Flotation Tests on Zinc-Iron Middlings from the British Metals Corporation
 Head sample: Zn. 24.54 per cent. Pb. 2.51 per cent. Fe. 25.02 per cent. Au. 0.10 oz. per ton. Ag. 4.56 oz. per ton.

Test No.	Product	Weight		Analysis					Analysis per cent x weight per cent				Per cent of values			
		Grms	Per cent	Zn. per cent	Pb. per cent	Fe. per cent	Au. oz. per ton	Ag. oz. per ton	Zn.	Pb.	Au.	Ag.	Zn.	Pb.	Au.	Ag.
Mag-net-ic	Zinc.....	1796.5	57.4	38.09	2.67	7.90	0.03	3.87	2185	153.4	1.72	222.0	86.7	68.6	50.3	49.7
	Iron.....	1332.5	42.6	7.87	1.61	48.60	0.04	5.28	335	70.0	1.70	225.0	13.3	31.5	49.7	50.3
1	Zn. concentrate table.....	338.2	33.8	46.73	2.80	9.54	0.01	6.14	1580	94.6	1.35	207.5	65.1	36.3	33.9	39.1
	Pb. ".....	11.2	1.1	13.46	51.65	0.48	58.56	14.8	14.8	56.8	0.53	61.4	0.6	21.7	13.3	12.1
	Zn. middlings flotation.....	54.6	5.5	22.52	4.6	25.8	0.06	9.51	124	25.3	0.33	52.4	5.1	9.7	8.2	9.9
	Zn. tailings flotation.....	593.8	59.4	11.95	1.42	33.56	0.03	3.49	710	81.3	1.78	207.3	29.2	32.3	44.6	38.9
2	Concentrate.....	467.2	52.4	40.22	4.35	2120	228.0	81.0	81.8
	Middling.....	115.2	12.9	22.73	1.90	293	24.5	11.2	8.8
	Tailing.....	308.4	34.6	5.83	0.76	tr.	2.02	202	26.3	69.9	7.7	9.4
3	Concentrate.....	284	27.8	40.47	4.38	1125	122.0	46.2	49.4
	Middling.....	90.7	8.9	21.44	2.48	218	22.0	8.9	8.9
	Tailing.....	645.8	63.3	17.25	1.63	1080	103.0	44.9	41.7
	Pb. concentrate.....	17.5	1.7	6.22	40.95	0.60	49.4	106	69.6	1.02	70.4	4.3	31.2
4	Zn. ".....	448.7	45.2	37.02	2.37	1673	107	68.4	48.0
	Zn. middling.....	207.0	20.9	21.72	1.44	451.5	30.1	18.6	13.5
	Zn. tailing.....	318.5	32.2	6.51	0.50	209.5	16.1	8.6	7.3
	Concentrate.....	604.4	40.0	31.28	5.1	16.58	0.10	8.78	1372	204.0	4.0	351.0	57.2	78.2	58.0	72.8
5	Middling.....	253.2	16.8	25.34	1.7	24.20	0.07	3.60	426	28.5	1.2	60.5	17.8	11.0	17.4	12.5
	Tailing.....	652.0	43.2	13.88	0.65	27.97	0.01	1.64	600	28.2	1.7	70.8	25.0	10.8	24.6	14.7
	Concentrate.....	364.4	47.61	3.74	10.68	1742	67.6
10	Middling.....	89.9	25.48	4.72	26.07	229	4.72	26.07	8.9
	Tailing.....	541.2	11.15	1.17	37.53	607	23.5
	Concentrate.....	237.7	23.8	37.41	890	36.6
11	Middling.....	173.0	17.5	25.38	444	18.2
	Tailing.....	581.5	58.7	18.75	1101	45.2
	Concentrate.....	431.0	43.5	29.22	1271	51.5
13	Middling.....	221.2	22.3	28.64	639	25.9
	Tailing.....	339.7	34.2	16.34	559	22.6

III

REPORT ON WORK AND INVESTIGATIONS OF THE CHEMICAL
LABORATORY

H. C. Mabee

Chemist in Charge

The work covered by the staff of the chemical laboratory of the Division during the year 1923 may be included in the following classes:—

I. The regular routine analytical work in connexion with the investigations conducted on ores, etc., in the ore testing and research laboratories of the Division of Ore Dressing and Metallurgy.

II. Special investigations entailing preliminary chemical tests on ores and minerals with the object of detecting and regulating any interfering elements in the processes of concentration.

III. (a) A study of the results of the application of the flotation process to the treatment of low-grade nickeliferous pyrrhotite ores, with respect, more particularly, to the behaviour of the precious metal content and to the extent of the successful elimination of barren pyrrhotite in the tailings.

(b) A study of the application of leaching and electro-deposition processes for the treatment of Canadian pyrrhotite and pyrite ores.

Various interruptions of the work of the chemical laboratory occurred during the year due to changes in the staff, rearrangements and additions to the present laboratory buildings, etc. Notwithstanding this, however, more work was accomplished than in any previous year.

I. ROUTINE ANALYTICAL WORK

H. C. Mabee

The regular routine work was of the usual wide variety, covering 1,295 samples, involving about 4,800 chemical determinations, the results of which appear in the general reports on investigations conducted on ores and mineral products in the ore testing and research laboratories of the Division.

These samples may be included under the following classification:—

9 samples of antimony ores			1 samples of lead-zinc-manganese ore		
58	"	arsenic ores	55	"	molybdenite ores
121	"	copper ores	20	"	platinum-gold ores
500	"	gold ores	22	"	silver ores
24	"	gold-cobalt ores	53	"	silver-lead-zinc ores
90	"	gold-copper ores	193	"	silver-lead-zinc-iron
56	"	gold-silver-copper	55	"	silver residues
1	"	iron ore	36	"	graphite ores
18	"	for identification			

The renewed attention to gold mining continued during the year, and as a result, about seventy-five per cent of the total samples reported included results in gold and silver content. These ores were of various composition as the classification indicates, each variety requiring its special treatment. In order, therefore, to keep up with the work promptly, it was found necessary to purchase an additional muffle furnace and otherwise add to the assay laboratory equipment.

As the room provided for this work was too small to accommodate the additional equipment, and as it had become necessary to provide more space in which to carry on the work connected with the electro-deposition of iron, a request for an addition of sixty feet to the present temporary building was made. At the close of the year the work of construction was well under way.

II. SPECIAL INVESTIGATIONS PERTAINING TO THE INVESTIGATIONS CARRIED ON BY THE ENGINEERING STAFF

THE VALUE OF GIVING THE SILVER RESIDUES FROM THE DOMINION REDUCTION CO., COBALT, ONT., A WEAK SULPHURIC ACID WASH PREVIOUS TO CYANIDING, BY B. P. COYNE

Tests were made to find the value of a weak sulphuric acid wash previous to cyaniding. In each test, ore ground to 65 mesh was used.

No. 1, 50 grammes ore treated with	100 c.c. 1 per cent H_2SO_4 (approximately)
No. 2, 50 " " " "	100 c.c. 2 " " "
No. 3, 50 " " " "	100 c.c. 3 " " "

In each case, the sample was agitated for about 1 hour, the solution filtered off, 10 c.c. taken for analysis, the remainder made up to 100 c.c. with the original acid and then added to 50 grammes fresh ore, and the process repeated to find the amount of the acid consumed. The ore from the first treatment was washed until free from soluble iron and sulphate salts, the sample which required the least amount of washing was then cyanided.

<i>Iron dissolved (as Fe_2O_3)</i>	Grammes		
	1st	2nd	3rd
1 per cent acid, 50 grammes ore.....	0.0170	0.0255	0.0230
2 " " 50 " "	0.0425	0.0580	0.0570
3 " " 50 " "	0.0380	0.0670	0.0750

<i>Acid consumption—</i>	Per cent		
	Per cent	Per cent	Per cent
1 per cent acid, 50 grammes ore.....	44.8	10.4	3.4
2 " " 50 " "	37.9	25.4	2.9
3 " " 50 " "	36.5	30.7	0.1

After washing five times, sample No. 1 was found to be free from soluble iron and sulphates, samples Nos. 2 and 3 still held these soluble salts. Sample No. 1 was cyanided, using 200 c.c. 0.125 per cent KCN solution and 0.1 gramme lime per 50 grammes ore. The cyanide consumption was 9.2 lbs. per ton. It was found that a 1 per cent sulphuric acid wash was as efficient as a stronger acid wash for removing the soluble salts from the ore; that it was easier to wash out the soluble salts when 1 per cent acid was used; that this treatment reduces the cyanide consumption.

DETERMINATION OF SOLUBLE CYANIDES IN CRANBERRY HEAD GOLD ORE AND THE EFFECT OF CERTAIN REAGENTS ON CYANIDE CONSUMPTION, BY B. P. COYNE

Tests were made to determine whether the ore contained soluble cyanides, and to determine the effect of certain reagents on the cyanide consumption.

Test No. 1.—To determine whether the ore contained soluble cyanides 100 grammes ore (—65 mesh) were agitated for 24 hours with 600 c.c. water, 500 c.c. filtered off, 0.625 gramme KCN added to the filtrate and again agitated for 16 hours. Filtrate contained 0.124 per cent KCN.

Test No. 2.—Similar to No. 1 except that 1 gramme of lime was added to the ore.

Test No. 3.—100 grammes ore cyanided without the use of lime.

Test No. 4.—Similar to No. 2 except that 0.5 gramme NaOH was added instead of using the 1 gramme lime.

Test No. 5.—100 grammes ore cyanided with 0.125 per cent KCN and 0.15 gramme Na_2O_2 .

It was found that the ore does not contain any cyanicides soluble under working conditions; that the use of Na_2O_2 reduced the cyanide consumption to 1.56 lbs. per ton ore.

DETERMINATION OF SOLUBLE SALTS IN EUSTIS COPPER ORE, AND THE EFFECT OF ADDING LIME FOR SELECTIVE FLOTATION OF THE CHALCOPYRITE, BY B. P. COYNE

A number of tests were made to determine the amount of soluble salts in the ore and the effect of adding lime. In each case 1,000 grammes ore were agitated with 1,000 c.c. distilled water. The time of contact varied from 35 to 45 minutes. In test No. 5, 5 grammes of lime were added to the pulp.

SOLUBLE SALTS

Test No.	35 minutes contact		45 minutes contact	
	Fe. grms. per litre	SO_3 grms. per litre	Fe. grms. per litre	SO_3 grms. per litre
1.....	0.073	0.23	0.072	0.20
2.....	0.024	0.53	0.020	0.53
3.....	2.09	9.39	2.32	9.67
4.....	0.073	0.23	0.072	0.20
5.....	trace	0.43	trace	0.28

It was found that the oxidized ore (test No. 3) contained a much greater amount of soluble salts than the unoxidized ore; that the addition of lime kept the iron from going into solution.

III. A STUDY OF THE RESULTS OF TWO SMELTING TESTS ON SHEBANDOWAN LAKE ORE, AND OF THE ELIMINATION OF BARREN PYRRHOTITE IN THE FLOTATION TAILINGS FROM THE SUDBURY ORES.

(a) THE PRECIOUS METAL VALUES IN PRODUCTS FROM SMELTING TESTS ON COPPER-NICKEL ORE OF SHEBANDOWAN LAKE, ONT., BY H. C. MABEE

In the Summary Report of the Mines Branch for 1922 there is given, under test No. 172, the assay results of five samples of ore from the Shebandowan Lake nickel-copper deposits, collected by T. L. Tanton, and submitted to the Ore Dressing Laboratories through the Director of the Geological Survey. In January, 1923, a further shipment of 400 lbs. of this ore was received from Messrs. Jamieson and Peacock, Duluth, Minn., upon which the following experimental tests were conducted:—

Two lots of twenty pounds each were fluxed and smelted in large pots and a low-grade matte was obtained. Each of these lots was ground separately, the iron was partly eliminated by treatment with dilute hydro-

chloric acid and the residue carefully roasted was fluxed and assayed in the usual manner for the platinum metal content, with results as shown below:—

Analysis of Ore

Nickel.....	2.97 per cent	Lime.....	2.50 per cent
Copper.....	1.65 “	Magnesia.....	7.64 “
Cobalt.....	0.15 “	Sulphur.....	15.84 “
Iron.....	24.30 “	Gold.....	0.01 oz. per ton
Alumina.....	10.15 “	Platinum.....	0.03 “
Silica.....	26.55 “	Palladium.....	0.048 “

From the above analysis the content of the ore may be calculated as follows:—

Total sulphides—Pentlandite.....	14.22 per cent
Pyrrhotite.....	23.24 “
Chalcopyrite.....	4.71 “
	42.18 per cent
Gangue.....	57.82 “

The average analysis of the slag from these two smelting tests on the ore is as follows:—

Nickel and cobalt.....	0.06 per cent
Copper.....	0.10 “
Iron.....	7.57 “
Sulphur.....	0.51 “

The weight of matte obtained averaged about 37.5 per cent of the original weight of ore in the charge. Taking into consideration the analysis of the slag as indicated above, it will be observed that only about 1 per cent of the total sulphides, including pyrrhotite, was eliminated in the slag, hence a yield of low-grade matte. This may be accounted for in this case, as these smelting operations were conducted almost out of contact with the air and consequently less opportunity for oxidation than would exist in regular smelting operations. The analysis of the matte gave the following result in platinum group metals:—

Melt No. 1.—Gold, 0.027 oz. per ton. Platinum, 0.080 oz. per ton. Palladium, 0.128 oz. per ton.
Melt No. 2. “ 0.030 “ “ 0.085 “ “ 0.131 “

THE ELIMINATION OF BARREN PYRRHOTITE BY CONCENTRATION OF THE
NICKELIFEROUS PYRRHOTITE ORES OF THE SUDBURY DISTRICT BY
SELECTIVE FLOTATION, BY H. C. MABEE

In continuation of the experimental tests by flotation made during 1922 on the Sudbury pyrrhotite ores, further investigations were carried on with respect to the applications of this process, with regard more particularly to the nature of each flotation product, the behaviour of each of the sulphides contained, and the relative losses of precious metal values in the tailing product. In this work several analyses were made, especially of the tailing products, and from these the nature of their content was determined. The following shows the analyses of the two lots of ore upon which the tests were performed:—

Ore	Copper per cent	Nickel per cent	Iron per cent	Sulphur per cent	Silica per cent	Oz. per ton		
						Au.	Plat.	Palad.
No. 3 mine, International Nickel Company.....	1.25	1.35	23.7	10.7	37.85	0.008	0.014	0.017
Murray mine, British Amer- ica Nickel Company.....	0.47	1.17	20.9	11.19	42.15	0.005	0.0067	0.009

From these analyses it may be calculated that the content of each of the above samples is made up as follows:—

Ore	Pentlandite per cent	Chalcopyrite per cent	Pyrrhotite per cent	Gangue per cent	Total sulphides
No. 3 mine, International Nickel Company.....	6.14	3.57	18.56	71.73	28.27
Murray mine, British America Nickel Company.....	4.59	1.34	24.44	69.63	30.37

In previous Summary Reports of the Mines Branch, a complete report of the results obtained from the application of flotation to the ores will be found, embracing percentages of recovery, assays, etc., of each of the flotation products. Upon further examination of the tailings derived from these tests, the content proved to be the following on analysis:—

Tailing from	Copper per cent	Nickel per cent	Iron per cent	Sulphur per cent	Silica per cent	Oz. per ton		
						Gold	Plat.	Palad.
No. 3 mine, International Nickel Company.....	0.05	0.16	8.14	3.23	46.75	0.002	0.0027	0.0038
Murray mine, British America Nickel Company.....	0.05	0.21	9.49	5.52	41.30	0.0009	0.0011	0.0026

From these results the composition of the tailings may be assumed as follows:—

Tailing from	Pentlandite per cent	Chalcopyrite per cent	Pyrrhotite per cent	Gangue per cent	Total sulphides
No. 3 mine, International Nickel Company.....	0.73	0.14	7.62	91.51	8.49
Murray mine, British America Nickel Company.....	0.96	0.14	13.12	85.78	14.22

Considering the pyrrhotite composition of the ores, namely, 18.56 per cent and 24.44 per cent respectively, and of the tailings, namely 7.62 per cent and 13.12 per cent respectively, 30 per cent of the pyrrhotite content in the first ore, and 41 per cent of the pyrrhotite content of the second ore, has been eliminated in the tailings. This elimination of almost barren pyrrhotite, together with the gangue minerals, means considerable saving in several directions.

With regard to the particular sulphide minerals with which the platinum group metals are associated nothing definite has been determined. In studying the ratios between these metals and the copper, nickel, iron and sulphur content of the flotation products, there is no definite uniformity of values or proportion between these metals and the chalcopyrite, pentlandite and pyrrhotite content in the ores and flotation products. Whether this should be accepted in support of the opinion that the lower grade ores carry relatively higher values in the platinum group metals is questionable.

- (b) THE HYDROMETALLURGICAL TREATMENT OF PYRRHOTITE AND PYRITE ORES SUCH AS THE LEACHING OF PYRRHOTITE AND THE ROASTING AND LEACHING OF PYRITE, WITH SUBSEQUENT PRODUCTION OF IRON BY ELECTRO-DEPOSITION AND THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS, BY R. J. TRAILL

Introductory.—The process being investigated was described in the Summary Report for 1922, and is known as the Eustis process.

Initial laboratory scale experiments were reported last year, and these have been continued, with the object of securing more complete data regarding the suitability of the process for the treatment of iron sulphide ores, preparatory to a study on a larger scale.

The satisfactory results obtained from these small laboratory tests show that the process gives promise of being commercially applied to the treatment of Canadian pyrrhotite and pyrite ores, and warrants the continuation of the investigation on a larger scale with a complete cycle of operations, including the electro-deposition of the iron. One of the greatest difficulties will be to obtain solutions from the ores, free from impurities detrimental to electro-deposition.

The production of iron by electrolytic methods from the leaching of sulphide ores is attracting increased interest, particularly in England and the United States. The Department has had various enquiries relative to the subject under investigation.

General Procedure—

Leaching Apparatus

An enamel pail of 5 to 6 litres capacity with a glass agitator and an asbestos slate cover was found best for leaching purposes. The solution was heated by means of a steam jacket around the pail.

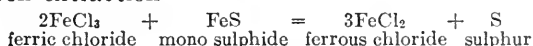
Solutions for leaching were prepared from ordinary grade ferric chloride containing about 20 per cent iron, and the ferrous chloride, purified from copper and lead, obtained in previous leaching tests.

Treatment of Pyrrhotite

The ore used, in the following tests, was a shipment of pyrrhotite received from the Smith property on the west side of lake Memphremagog, Que., and contained about 80 per cent pyrrhotite and 2.5 per cent chalcopyrite. The general procedure was to treat, by continuous agitation, a certain quantity of the ore, ground to pass 200 mesh, with a solution containing known quantities of ferric and ferrous chlorides, at a temperature from 95° to 98° C. Samples of the solution were drawn off at various intervals to ascertain the extent of the reaction between the ore and the ferric chloride. When reaction was complete, or the set time limit had expired, the leach was cooled, filtered, and the volume noted or made up to the original bulk. The iron content (ferrous and ferric) was ascertained, and the residue dried, weighed and assayed for iron, copper, etc.

Calculation of Extraction

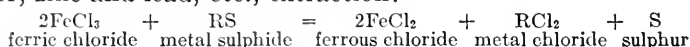
Iron extraction—



From the amount of ferrous iron in grammes found after leaching, subtract the amount of ferrous iron in grammes originally present in the

leach solution, plus twice the amount in grammes of copper, zinc or lead dissolved (obtained by subtracting amount in residue from amount in ore charge). The result gives the amount of ferrous iron formed in the reaction, and this divided by 3 gives amount of iron dissolved from the ore charge. Percentage extraction is arrived at by dividing the amount of dissolved iron by amount of iron treated and multiplying by 100.

Copper, zinc and lead, etc., extraction:—



Extractions of these metals are figured by subtracting the amount found in the residue from the amount contained in the ore charge. This amount divided by the amount in ore charge multiplied by 100 equals percentage.

Test No. 181

ORE—SMITH PYRRHOTITE

Leaching test No. 5—

Purpose of test.—This test was made to study effect of leaching with a solution containing both ferric and ferrous chlorides, the condition that will occur in regular practice.

Quantities—

3.6 litres solution—	82	grammes per litre ferrous iron =	295.2	grammes Fe.
	71	“ ferric “ =	250.6	“
25S grammes ore —	49.6	per cent iron	= 128.0	“
	0.85	“ copper	= 2.2	“
	1.14	“ zinc	= 2.94	“ Cu. Zn.

Excess of ore over theoretical required = 4.0 per cent.

The solution was heated to 70° C., the ore added, and temperature raised and maintained at 95° to 98° C.

Test taken after 30 minutes leaching showed	119	grammes per litre ferrous Fe.
“ “ 1 hour	136	“
“ “ 3 hours	160	“
“ “ 5 “	172	“
“ “ 7 “	178	“
“ “ 8 “	180	“

Test stopped at this period, cooled and filtered.

Volume 3.3 litres at 180 grammes per litre ferrous Fe =	594	grammes Fe.
and 7 “ ferric Fe. =	23.1	“

Residue weighed 140 grammes, and contained:—

Fe.....	25.8	per cent =	36.1	grammes
Cu.....	0.35	“ =	0.49	“

Result of 8 hours extraction:—

Iron extraction.....	75.4	per cent
Copper “.....	77.7	“

This result shows that the presence of ferrous chloride in addition to the ferric chloride does not materially affect the leaching action.

Leaching test No. 8—

Purpose.—In depositing iron from a solution of ferrous chloride, a certain amount of ferric chloride is formed (compare Summary Report,

1922). In the process this ferric chloride, with some ferrous chloride, will be conveyed from the cell to the leaching apparatus. The leaching process on large scale operation may be conducted in two ways. (1) An excess of ore may be maintained in the leaching agitator, so that the ferric solution will be completely reduced in the one operation, and the residue obtained, treated for recovery of sulphur by distillation, or a non-oxidizing roast. The product from this roast would be available for further leaching treatment and recovery of iron. (2) Leaching may be done by a counter-current extraction method. In this method two or more leaching agitators would be used, designated primary and secondary. The solution coming from the cell would be fed to the primary agitator, where the ore already partly treated in the secondary agitator and fed at intervals to the primary would be acted on, the solution becoming reduced to a certain degree, and finally being completely reduced in the secondary agitator by addition of fresh ore.

To study the feasibility of the latter method the following test was run, leaching fresh ore first with a solution considerably reduced and then leaching the residue with a solution of a composition similar to that coming from the electrolytic cell.

Quantities—

Solution No. 1 contained—			
3.6 litres at	132 grammes per litre	ferrous Fe. = 475.2	grammes Fe.
20	"	ferric Fe. = 72.0	"
Solution No. 2 contained—			
3.6 litres at	71 grammes per litre	ferrous Fe. = 255.6	grammes Fe.
63	"	ferric Fe. = 226.8	"
Ore charge—			
300 grammes at	49.6 per cent Fe.	= 148.8	grammes Fe.
	0.85 " Cu.	= 2.55	"
	1.14 " Zn.	= 3.42	"
Excess ore = 3.5 per cent.			

The ore was treated with solution No. 1 at 95° C. for one hour, after which time reduction was complete. Solution was cooled and filtered.

Volume 3.37 litres at 165 grammes per litre ferrous Fe = 556 grammes Fe. The residue without drying was treated with solution No. 2 at 95 to 98° C. and the time test showed:—

After 1 hour leaching.....	113 grammes per litre ferrous Fe.
" 3 hours "	134 " "
" 6 " "	148 " "
" 8 " "	160 " "
" 10 " "	172 " "
" 12 " "	179 " "

The test was stopped at this period, cooled and filtered.

Volume 3.15 litres at	179 grammes per litre ferrous Fe. = 563.8	grammes Fe.
and at 2	"	ferric Fe. = 6.3

The residue weighed 145 grammes and contained:—

Fe.....	20 per cent = 29 grammes
Cu.....	0.48 " = 0.696 "

Result, 13 hours extraction:—

Fe extraction.....	82.6 per cent
Cu. "	72.7 "

From this test it is evident that this method of treatment does not affect the percentage extraction. It may be observed, however, that the action is somewhat slower, due possibly to the sulphur that is set free, adhering to or coating particles of the ore.

The question whether method No. 2, or the straight single treatment method No. 1 would be the more satisfactory and economical in general practice will be more carefully investigated when equipment of the continuous circuit type has been installed.

Leaching tests Nos. 9, 10 and 11—

Purpose.—In various leaching tests it has been observed that the copper extraction has varied from 50 to 85 per cent, and an explanation has been sought for this difference in extraction.

In the leaching of chalcopyrite with ferric chloride, it is possible that both cupric and cuprous chloride may be formed. Cuprous chloride, it is claimed, when boiled with freshly precipitated sulphur, deposits the copper as cuprous sulphide, Cu_2S . It is quite possible that this precipitated sulphide of copper is insoluble in the conditions obtaining in the leaching. This explanation is borne out to some extent by the results obtained in the following tests:—

Leaching test No. 9—

Quantities—

4 litres solution at 70 grammes per litre	ferrous Fe. = 280	grammes Fe.
68 " "	ferric Fe. = 272	" "
275 grammes ore at 49.6 per cent Fe.	= 136.4	" "
0.85 " Cu.	= 2.34	" Cu.
1.14 " Zn.	= 3.13	" Zn.
Excess ore = 4.3 per cent.		

The solution was heated to 75° C., the ore added, and the temperature increased and maintained at from 95 to 98° C. Time tests taken showed:—

After 1 hour.....	135 grammes per litre	ferrous Fe
" 3 hours.....	156	" "
" 5 ".....	175	" "
" 6 ".....	180	" "
		(volume about 3.4 litres)

Test was stopped at this period, solution cooled rapidly and filtered immediately, the residue being slightly washed.

Volume 3.75 litres at 163 grammes per litre	ferrous Fe. = 611.25	grammes Fe.
13 " "	ferric Fe. = 48.75	" "

Residue weighed 135 grammes and contained:—

Fe.....	21.7	per cent = 29.3	grammes
Cu.....	0.25	" = 0.34	"

Result 6 hours extraction:—

Fe extraction.....	79.4	per cent
Cu extraction.....	85.5	"

Leaching test No. 10—

Purpose.—To duplicate test No. 9.

Quantities—

4 litres solution at 69 grammes per litre	ferrous Fe. = 276	grammes Fe.
73 " "	ferric Fe. = 292	" "
280 grammes ore at 49.6 per cent	Fe. = 138.8	" "
0.85 " Cu.	= 2.38	" Cu.
1.14 " Zn.	= 3.192	" Zn.
No excess ore.		

The solution was heated to 75° C., the ore added and temperature raised to 95–98° C. Time tests taken showed:—

After 1 hour.....	133	grammes per litre ferrous Fe.
" 2 hours.....	156	" "
" 3 "	163	" "
" 4 "	172	" "
" 5 "	179	" "
" 6 "	195	" " (volume about 3.2 litres)

Test stopped at this period, cooled rapidly and filtered immediately washing residue.

Volume 3.75 litres at 166 grammes per litre ferrous Fe. = 622.5 grammes Fe.
9 " ferric Fe. = 33.75 "

Residue weighed 140 grammes and contained:—

Fe.....	25.2	per cent =	35.2	grammes
Cu.....	0.23	" =	0.32	"

Result 6 hours extraction:—

Fe. extraction.....	80.7 per cent
Cu. extraction.....	86.5 "

This result agrees very closely with that obtained in test No. 9 in so far as extraction is concerned.

Leaching test No. 11—

Purpose.—To find copper extraction under conditions that will allow precipitated sulphur to act on any cuprous chloride formed in leaching reaction.

Quantities used—

4 litres at	70 grammes	per litre	ferrous	Fe.= 280 grammes	Fe.
at	72	"	ferric	Fe.= 238	" "
280 grammes ore at	49.6	per cent	Fe.	= 138.8	" "
	0.85	"	Cu.	= 2.38	" Cu.
	1.14	"	Zn.	= 3.19	" Zn.
Excess ore = 0.25 per cent.					

Excess ore = 0.25 per cent.

Solution heated to 75° C., ore added and temperature raised and maintained from 95 to 98° C. Time tests taken showed:—

After 1 hour.....	136	grammes per litre ferrous Fe.
“ 2 hours.....	154	“ “
“ 3 “	166	“ “
“ 4 “	176	“ “
“ 6 “	192	“ “ (volume about 3.1 litres)

Test stopped at this period, allowed to cool slowly with agitation and let stand overnight. Solution filtered and residue washed. Extra washing required on account of calcium sulphate crystallizing.

Volume 4 litres at 152 grammes per litre ferrous Fe.= 608 grammes Fe.
at 13 " ferric Fe.= 52 "

Residue weighed 142 grammes and contained:—

Fe.....	24.1 per cent =	34.2 grammes
Cu.....	0.5 “ =	0.71 “

Result 6 hours extraction:—

Fe. extraction.....	76.3 per cent
Cu. extraction.....	70.1 "

Comparing the results of extraction on these tests it will be observed that test No. 11 shows a slightly lower iron extraction (due in part possibly to slight oxidation occurring in standing overnight) and it shows a very much lower extraction in copper amounting to 15 or 16 per cent. It would appear therefore that some such action, as suggested under purpose of tests Nos. 9, 10, and 11, has taken place.

COPPER RECOVERY TESTS

The solutions obtained from leaching tests 9 and 10 were treated for recovery of copper by the iron precipitation method, the procedure being as follows:—

The solution, contained in the enamel pail, was heated by means of a steam jacket to 90° C., and steel in the form of short turnings added. The amount of steel used was about 50 per cent in excess of that required to reduce the ferric iron present in the solution, and to precipitate the copper. After treatment the residue was filtered off, dried, and assayed for copper content.

Copper recovery from solution test No. 9.—3.75 litres solution containing 2 grammes copper was treated for 1½ hours at 90°C. with 40 grammes steel turnings. A residue weighing 25 grammes (mostly steel) was obtained containing 1.94 grammes copper. This would be equivalent to 97 per cent of the copper present in solution. Result:—

Copper recovery from solution or.....	97.0 per cent
Copper recovery from ore.....	83.0 “

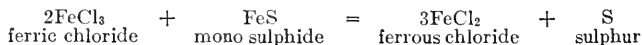
Copper recovery from solution test No. 10.—Solution from test No. 10 was treated in the same manner for 4 hours and showed a

Copper recovery from solution or.....	98.7 per cent
Copper recovery from ore.....	85.4 “

No attempt was made to rid the above solutions of the last trace of copper, as it seems reasonable to assume that this would be accomplished in a second treatment with iron or steel.

RECOVERY OF SULPHUR

The process as described in the Summary Report for 1922 provides for the recovery of the sulphur. The reaction between the ore and the ferric chloride is shown as—



the sulphur set free being found in the gangue or insoluble material. It is proposed to recover this sulphur by distillation.

In addition to this, where the single extraction method, as suggested in leaching test No. 8, is used, a product suitable for further leaching may be obtained after the sulphur distillation, thereby recovering the iron and other metals in the residue.

For the purpose of determining the feasibility of this idea the following test was made.

The iron content of these residues runs from 22 to 25 per cent, consisting of unattacked pyrrhotite or pyrite (if present in ore), iron as oxy-chloride precipitated by hydrolysis, and perhaps iron precipitated by lime or magnesia in the ore.

The copper content ranges from 0.23 to 0.5 per cent, present as unaltered chalcopyrite or chalcocite and possibly precipitated cuprous sulphide. A retort type of container was made from a piece of iron pipe 8 inches long by 2 inches inside diameter, closed at one end with a screw cap, and the other end fitted with a screw cap having a three-eighth-inch diameter pipe as outlet, bent at right angles.

The retort containing the charge was placed in a small electric muffle furnace at 750°C. for 2½ hours. The retort was then removed from the furnace, allowed to cool, and the loss in weight determined. No special precautions were taken to assure the complete condensation of the gaseous sulphur, the outlet simply delivering into a loosely covered receiver and the pipe being heated by means of a Bunsen burner to prevent sulphur condensing in it.

The residues obtained from tests 9, 10 and 11 were thus treated and showed volatile losses respectively as follows:—

No. 9.....	55 per cent loss
No. 10.....	53 " "
No. 11.....	52 " "

From residues Nos. 10 and 11, 42 grammes and 35 grammes of sulphur were recovered, equivalent to about 60 and 52 per cent respectively. Much sulphur was lost because of the unsuitability of the condensing apparatus, and some sulphur would be oxidized to sulphur dioxide (SO₂) in the initial stage of heating. The sulphur recovered appeared to be of quite good grade, judging from its colour and general appearance.

The test satisfactorily demonstrated the possibility of recovering the sulphur from these residues. The material left, after the sulphur distillation, had the following composition:—

		Iron	Copper	Sulphur	SO ₃
Residue test	No. 9....	47.30 per cent	0.54 per cent	28.16 per cent	1.61 per cent
"	No. 10....	52.70 "	0.52 "	30.90 "	0.15 "
"	No. 11....	52.00 "	1.00 "		

There was insufficient of this material to make a standard comparative leaching test, but a treatment with ferric chloride proved that this material was as easily leached as the natural pyrrhotite, and that recoveries equivalent to those obtained on the raw ore could be expected.

Test No. 182

TREATMENT OF PYRITE

Pyrite is much less soluble in ferric chloride than is pyrrhotite, but it is readily made soluble by a simple non-oxidizing roast. By this treatment an artificial pyrrhotite is formed, one atom of sulphur being driven off, which, by a suitable apparatus, may be recovered as a by-product.

A test illustrating the roasting of pyrite and subsequent leaching was described in Summary Report for 1922 under test No. 182.

Further tests have been carried out on this method of treatment and are described below. The retort type of container described as used in sulphur recovery from pyrrhotite residue treatment tests Nos. 9, 10 and 11, was used in the roasting process.

Two distinct methods were used and compared in effecting the elimination of the sulphur atom. (A) The first method was in the nature of a reducing roast. The charge, placed in the 8 by 2-inch tube, closed at one end, was subjected to a temperature of 750°C. in an electric muffle furnace. Almost immediately sulphur was set free, which combined with the oxygen in the container forming a reducing gas, namely, sulphur dioxide (SO₂). The gaseous sulphur gradually replaced the SO₂ in the container and burned at the outlet. This feature served as an indication of the progress of the roasting, the flame gradually diminishing as the roasting progressed. When the flame died out the container was withdrawn from the furnace and placed on end, the open end being loosely covered to prevent undue oxidation. It was observed that the product was strongly magnetic. Several samples weighing about 200 grammes were roasted in this manner for a period of 5 to 7 hours and results were fairly concurrent, the magnetic portion (using a horseshoe magnet) running from 88 to 92 per cent, and the total iron 50 to 52 per cent.

(B) The second method employed was by distillation, and is identical with that described in the treatment of residues from pyrrhotite leaching tests Nos. 9, 10 and 11, the container being similar to the one used in the previous method. This container was charged with 200 grammes ore and placed in the muffle furnace at 750°C. It was found that the time necessary to accomplish the elimination of the sulphur atom required from 12 to 14 hours. This time was reduced to about 7 to 9 hours by increasing the temperature to 840°C. No special precautions were taken to condense the gaseous sulphur, a simple air-cooled receiver with a loose cover being employed. This served to recover only 30 to 45 per cent of the sulphur in the ore charge. The sulphur so obtained was of a deep yellow colour, and there was also a very small quantity of dark brown oily fluid which finally solidified with a deep yellow colour.

No satisfactory method of observing the progress of distillation was available so that it was necessary to examine the residue for unaltered pyrite, and if any was found the whole was re-distilled. A number of samples were thus treated, and on examination the residues were non-magnetic, and of a slightly lighter colour than the magnetic material.

To determine the difference between the magnetic and non-magnetic products an analysis was made. The samples were first passed over an Ullrich magnetic separator. From the material obtained by method A, 98 per cent was separated as magnetic, the 2 per cent non-magnetic being discarded. From the material obtained by method B, 14 per cent was separated as magnetic and discarded. Analysis of the magnetic and non-magnetic resulted as follows:—

	Magnetic	Non-magnetic
Total iron.....	53.50 per cent	51.85 per cent
Ferrous iron.....	52.25 "	51.85 "
Sulphur.....	32.90 "	33.40 "
Copper.....	1.95 "	1.90 "
Lead.....	0.76 "	0.74 "

It will be observed that the magnetic product contains a small percentage, namely, 1.25 per cent ferric iron, possibly due to magnetite having been formed in the roasting.

Leaching roasted pyrite.—Leaching tests were made on the magnetic and non-magnetic products to determine which product gave the best extractions, and to find how the extractions compared with those of pyrrhotite under the same conditions as in the pyrrhotite leaching tests Nos. 9 and 10.

Leaching test No. 13 on magnetic material—

Quantities:—

4 litres solution at 71	grammes per litre ferrous Fe. = 284	grammes Fe.
73	ferric Fe. = 292	"
265 grammes ore at 52.25 per cent available Fe.	= 138.5	"
1.95 " copper	= 5.17	" Cu.
0.76 " lead	= 2.0	" Pb.

No excess ore.

Solution heated to 75°C., ore added, and temperature raised to 95 to 98°C. Time tests taken showed:—

After 2 hours.....	127 grammes per litre ferrous Fe.
" 4 "	160 " "
" 6 "	173 " "

Test stopped at this period, rapidly cooled and filtered, washing residue and making up to original volume of 4 litres.

Volume 4 litres at 152 grammes per litre ferrous Fe. = 608 grammes Fe.
17 " ferric Fe. = 68 " "

Residue weighed 135 grammes and contained:—

Iron.....	22.2 per cent = 30 grammes
Copper.....	0.35 " = 0.47 "
Lead.....	Trace

Result 6 hours extraction:—

Iron extraction.....	75.0 per cent
Copper extraction.....	90.9 " "
Lead extraction.....	95 to 100 " "

Leaching test No. 13B:—

Purpose.—To duplicate test No. 13.

Quantities:—

4 litres solution at 73	grammes per litre ferrous Fe. = 292	grammes Fe.
70	ferric Fe. = 280	"
265 grammes ore at 52.25 per cent available Fe.	= 138.5	"
1.95 " copper	= 5.17	" Cu.
0.76 " lead	= 2.00	" Pb.

Excess ore, 4.0 per cent.

Solution heated to 75°C., ore added, and temperature raised to 95 to 98°C. Time tests taken showed:—

After 2 hours.....	125 grammes per litre ferrous Fe.
" 4 "	140 " "
" 6 "	163 " "

Test stopped at this period, cooled rapidly and filtered, washing residue and making up to 4 litres.

Volume 4 litres at 157 grammes per litre ferrous Fe. = 628 grammes Fe.
15 " ferric Fe. = 60 " "

Residue weighed 134.5 grammes and contained:—

Iron.....	26.45 per cent	= 35.6 grammes (90 per cent in ferrous condition)
Copper.....	0.45 "	= 0.6 "
Lead.....	Trace	

Result of 6 hours extraction:—

Iron extraction.....	77.7 per cent
Copper "	88.4 "
Lead "	95 to 100 "

Leaching test No. 14—Non-magnetic material—

Purpose.—Comparison with test No. 13.

Quantities:—

4 litres solution at 73	grammes per litre ferrous Fe. = 292	grammes Fe.
70	ferric Fe. = 280	"
270 grammes ore at 51.85 per cent iron	= 140	"
1.90 " copper	= 5.13	" Cu.
0.74 " lead	= 2.00	" Pb.
Excess ore, 5 per cent.		

Solution heated to 75°C., ore added and temperature raised to 95 to 98°C.
Time tests taken showed:—

After 2 hours.....	134 grammes per litre ferrous Fe.
" 4 "	156 " "
" 5 "	186 " "

Test stopped at this period, rapidly cooled and filtered, washing residue and making up to 4 litres.

Volume 4 litres at 168 grammes per litre ferrous Fe. = 672 grammes Fe.
3 " ferric Fe. = 12 "

Residue weighed 142.1 grammes and contained:—

Iron.....	21.15 per cent	= 30 grammes (88 per cent ferrous)
Copper.....	1.23 "	= 1.75 "
Lead.....	Trace	

Result 6 hours extraction:—

Iron extraction.....	88 per cent
Copper "	66 "
Lead "	95 to 100 "

Leaching test No. 14B—

Purpose.—Duplicate of 14.

Quantities:—

4 litres solution at 72	grammes per litre ferrous Fe. = 288	grammes Fe.
71	ferric Fe. = 284	"
270 grammes ore at 51.85 per cent iron	= 140	"
1.90 " copper	= 5.13	" Cu.
0.74 " lead	= 2.00	" Pb.
Excess ore, 3.6 per cent.		

Solution heated to 75°C., ore added and temperature raised to 95 to 98°C.
Time tests taken showed:—

After 2 hours.....	143 grammes per litre ferrous Fe.
" 4 "	186 " "
" 6 "	196 " "

Test stopped at this period, rapidly cooled and filtered, washing residue and making up to 4 litres.

Volume 4 litres at 166 grammes per litre ferrous Fe. = 664 grammes Fe.
8 " ferric Fe. = 32 "

Residue weighed 136.6 grammes and contained:—

Iron.....	18.3	per cent=25	grammes (91 per cent ferrous)
Copper.....	0.75	"	= 1.02 "
Lead.....	Trace		

Result 6 hours extraction:—

Iron extraction.....	86.6	per cent
Copper "	80.0	"
Lead "	95 to 100	"

Comparing these results it would appear that the magnetic product gives the higher copper extraction and the non-magnetic gives the higher iron extraction. No explanation can be given for the difference in copper extraction on duplicate tests Nos. 14 and 14B. The iron and copper extractions obtained on the magnetic material compare very closely with those obtained on the natural pyrrhotite, and the iron extraction from non-magnetic material is slightly better than from the natural pyrrhotite.

COPPER RECOVERY

95 to 98 per cent of the copper was recovered from the solutions by precipitating on steel turnings, but no lead was found to be precipitated with the copper.

TEST ON COMPLEX SULPHIDE ORE

In an endeavour to obtain some idea as to how other metallic sulphides, and particularly gold and silver, would be acted upon in this process, a preliminary test was run on a complex sulphide ore. It was thought advisable to use a sample with high gold and silver content, as more accurate data would probably be obtained than by using low-grade ore.

The sample, selected for this test, was from a concentrate obtained from the Woodsworth ore which was shipped to this laboratory for ore dressing tests. An analysis of the concentrate is as follows:—

Gold.....	20.2	ozs. per ton
Silver.....	15.68	"
Copper.....	0.92	per cent
Lead.....	2.50	"
Zinc.....	15.20	"
Iron.....	28.90	"
Sulphur.....	39.28	"
Silica.....	8.00	"
Arsenic.....	1.57	"

The iron was present mostly as pyrite, and the sample was therefore roasted in a non-oxidizing atmosphere as outlined in pyrite roasting treatment A, the temperature not exceeding 750°C. The roasted product had the following analysis:—

Gold.....	22.2	ozs. per ton
Silver.....	17.2	"
Copper.....	1.08	per cent
Lead.....	2.94	"
Zinc.....	17.70	"
Iron.....	34.0	"
Sulphur.....	29.20	"
Arsenic.....	0.57	"

The loss in weight was 14.6 per cent.

From these figures it would appear that the volatiles consisted only of sulphur and arsenic, 70 per cent of the latter being volatilized. Theoretically, the product contains 55.7 per cent of metallics capable of reacting with ferric chloride (excluding arsenic).

Leaching test.—A leaching test was made on this roasted material, observing the regular procedure.

Quantities:—

3.3 litres solution at 30	grammes per litre ferrous Fe. = 99	grammes Fe.
71	ferric Fe. = 234.3	"
210 grammes ore at 55.7	per cent reacting metals = 117	"
34	iron = 71.4	"
17.7	zinc = 37.17	grammes Zn.
2.94	lead = 6.17	Pb.
1.08	copper = 2.27	Cu.

No excess ore.

Solution heated to 75°C., ore added and temperature raised to 95 to 98°C.

Time tests taken showed:—

After 2 hours.....	91 grammes per litre ferrous Fe.
" 4 "	134 " "
" 5 "	143 " "
" 6 "	148 " "

Test was stopped at this period, cooled and filtered, and residue washed.

Volume 3 litres at 132.33 grammes per litre ferrous Fe.=397 grammes Fe. and trace ferric Fe.

Residue weighed 96 grammes and contained:—

Iron.....	6.46 per cent	=6.2	grammes
Copper.....	0.35 "	=0.336	"
Lead.....	Trace		
Zinc.....	1.97 "	=1.89	"
Gold.....	50.79 ozs. per ton	=0.1672	"
Silver.....	6.88 "	=0.02265	"

Result 6 hours extraction:—

Iron extraction.....	98.5 per cent
Copper "	85.2 "
Lead "	99.0 "
Zinc "	94.9 "
Gold "	None
Silver "	81.7 "

Arsenic was present in both residue and solution, but was not determined.

The extraction is remarkable in that it is about 98 per cent of the theoretical. The iron extraction is outstanding and it is perhaps possible that some of the other metals act in the manner of catalysts and increase the extraction. The gold is left almost free from other metals and from the residue it should easily be recovered.

RECOVERY OF METALS

The recovery of these metals from solution presents an interesting problem, upon which the success of this process will very much depend. It is essential that the method employed should not introduce any substances that would interfere in any way with the subsequent recovery of the iron by electro-deposition. Scrap iron or steel, hydrogen sulphide,

and sodium sulphide are probably the most commonly used precipitants. These precipitants were tried with the following results:—

Recovery using steel turnings

A quantity of the solution from the above test was treated with steel turnings for $2\frac{1}{2}$ hours with stirring, at a temperature of $75^{\circ}\text{C}.$, the precipitate filtered off and assayed. The following recoveries were made, based on the amount of the metal in solution:—

Copper.....	85.7 per cent
Lead.....	84.9 "
Silver.....	77.8 "
Zinc.....	None

It is quite within reason to expect that a second treatment would have resulted in a more complete recovery. The recoveries obtained above would represent a total recovery of these metals from the ore of:—

Copper.....	73.0 per cent
Lead.....	25.0 "
Silver.....	63.6 "

The silver recovery would of course be increased by the amount of silver recovered with the gold values in the subsequent treatment of the residue.

Recovery using sodium sulphide

The amount of sodium sulphide required to precipitate the reacting metals in solution, plus a small excess, was added to the solution at about $70^{\circ}\text{C}.$ and treated with occasional shaking for 30 minutes. The precipitate obtained was filtered off and assayed, showing the following recoveries of dissolved metals:—

Copper.....	64.5 per cent
Lead.....	25.0 "
Silver.....	48.5 "
Zinc.....	3.0 "

The filtrate was treated with a further quantity of sodium sulphide, using half as much as in the first treatment, in the same manner and time. This second treatment resulted in additional recoveries that brought the total recoveries up to:—

Copper.....	85.2 per cent
Lead.....	66.9 "
Silver.....	79.7 "
Zinc.....	10.5 "
Iron precipitated.....	1.7 "

A third treatment in the same manner showed a further increase in metals precipitated, resulting in total recoveries from solution of:—

Copper.....	88.5 per cent
Lead.....	88.0 "
Silver.....	87.0 "
Zinc.....	12.0 "

representing total recoveries from the ore of:—

Copper.....	75.4 per cent
Lead.....	88.0 "
Silver.....	71.0 "
Zinc.....	11.4 "

Recoveries using hydrogen sulphide

This test was conducted in a large flask, using hydrogen sulphide as precipitant under a slight pressure for 5 minutes. Precipitate was filtered off and an assay showed recoveries from solution of:—

Copper.....	92.0 per cent
Lead.....	8.6 “

Amount taken for this test, namely 200 c.c., was too small to permit accurate silver determinations, and therefore no recovery in silver could be figured. A second treatment with hydrogen sulphide for 20 minutes showed an increase that gave a total recovery from solution of:—

Copper.....	92.0 per cent
Lead.....	91.3 “

A third treatment with hydrogen sulphide for 20 minutes brought the total recovery from solution to:—

Copper.....	96.2 per cent
Lead.....	94.1 “

representing a total recovery from the ore of:—

Copper.....	82.0 per cent
Lead.....	94.1 “

This test clearly demonstrates the suitability of this process for the treatment of this type of ore, and it is evident that quite simple methods can be employed in the recovery of metals contained in the leach from this ore.

CONCLUSIONS

The tests were carried out on a laboratory experiment scale, anticipating, as far as possible, problems and conditions that would arise in actual practice, before attempting to experiment on a larger scale.

These tests have demonstrated that on an average 80 per cent of the iron and copper can be extracted with ferric chloride from pyrrhotite ores. Pyrite, with a preliminary treatment, has also been shown to be amenable to this treatment, with almost similar recoveries. Lead, zinc and silver are also easily extracted, gold is separated in a more concentrated condition, and methods for recovery of these metals have been described, with the exception of zinc, for which no satisfactory method of separation has yet been found.

The separation of iron has, so far, not been attempted, as no suitable apparatus is at hand. However, a special laboratory is being equipped with the necessary apparatus for electro-deposition of metals, and also with leaching and precipitating apparatus, so that a more complete study of the application of this process to Canadian ores can be made under conditions more closely approximating those of actual practice.

IV

SELECTIVE FLOTATION AS APPLIED TO CANADIAN ORES

C. S. Parsons

SELECTIVE FLOTATION PROBLEMS

For the purpose of this article, selective flotation is taken to mean the separation by flotation of one mineral from another, where two or more flotative minerals are present in an ore. The separations generally required are on complex ores, consisting of combinations of lead, zinc and iron sulphides, copper, zinc and iron sulphides, molybdenum and iron sulphides, antimony and arsenical iron sulphides, graphite and iron sulphides, and other similar combinations.

The logical way to separate these minerals by flotation is to float first the mineral which has the highest floating properties. A list of the more common minerals met with in selective flotation, in order of the ease with which they are floated, from the writer's own experience is: graphite, molybdenite, stibnite, galena, chalcopyrite, sphalerite, pentlandite, pyrite, arsenides and antimonides, etc., pyrrhotite, arsenical pyrite. For instance, in the selective flotation of a lead-zinc sulphide ore, the lead should be floated first, as it is the more easily floated mineral of the two, in a copper-zinc ore, the copper should be floated first, and so on.

OILS USED FOR SELECTIVE FLOTATION*

During the early stages in the development of the flotation process much more weight was given to the function of the oil than is given at present, as it was supposed that every ore required special oils. Dozens of different oils and oil mixtures were tested on each ore. Some mills were operated in which oil mixtures containing four or five different oils were used, whereas in all probability only one or two were essential. The Department has over 300 samples of oils, but of this number 90 per cent are never used in the regular test work. It has been found that satisfactory results can be obtained on the majority of ores from a group containing a few varieties. A list of the oils in this group is given, together with the selective properties of each as found by actual experience.

Reagents used for the flotation of lead sulphides in the presence of zinc and iron sulphides:—

Mixture of coal tar and coal tar creosote.

Cresylic acid.

Z cake or thiocarbanilide.

Steam distilled pine oil.

Light hardwood¹ creosote oil.

Reagents used for the flotation of zinc sulphides from iron sulphide after lead or copper sulphides have been removed:—

KK oil No. 2—Southwestern Engineering Co.

No. 634 creosote oil—Barrett Co.

Sodium creosote from hardwood.

YZ mixture, or xylidine and thiocarbanilide.

*Oil, in flotation terminology, is a generic term used to designate the organic substance that is used to produce a froth or effect selection of metalliferous minerals (Taggart).

No. 34 gravity fuel oil—G. W. Oil Co.

TT mixture.

Water-gas tar—Barretts.

Reagents used for the flotation of copper sulphides from zinc and iron sulphides:—

Mixture of coal tar and coal tar creosote.

XY mixture or xylidine and alpha-naphthylamine.

Neutral fractions of hardwood oils.

Reagents used for the flotation of copper sulphides from iron sulphides:—

Barretts No. 634.

XY mixture

TT mixture

Thiofizzan

ZY mixture

Sodium resinate—use very limited.

Fuel oils (paraffin base).

} Reagents of the General Engineering Co.

Reagents used for the flotation of copper-nickel sulphides from iron sulphides:—

Thiofizzan.

TT mixture.

XY mixture.

Fuel oil (paraffin base).

Coal tar and coal tar creosote mixtures.

Reagents used for the flotation of molybdenite and graphite from iron sulphides:—

Kerosene oil.

Frothing reagents used with the above:—

Steam distilled pine oil.

Fumol—manufactured by Canadian Electro Products Co.

Ketone oil—manufactured by the Standard Chemical Co.

ADDITION REAGENTS

The use of addition reagents is as important as the use of oils for selective flotation. If the conditions in the pulp are correct there is generally found to be considerable latitude with regard to the kind of oil used. In the writer's experience, the chief function of the more common addition reagents used in practice for selective flotation are:—

Lime.—The use of lime for the selective flotation of certain minerals was first brought to the attention of the Department in 1917, when operating a customs plant for the concentration of molybdenite ores from all points in the Dominion. A few carloads of very badly oxidized ore were received and trouble was encountered in floating the molybdenite. It was found that large quantities of soluble salts were contaminating the pulp. Lime was added to the ball mill with the object of preventing these salts from going into solution, and it was found that the lime, besides eliminating the trouble with soluble salts, also prevented the iron sulphides from floating, resulting in a much higher grade concentrate. In this connexion a very interesting phenomenon was observed. When the pulp was

contaminated with soluble sulphates, as much as 100 pounds per ton of kerosene oil could be added without causing any raw oil effect on the cells, a voluminous but absolutely barren froth being obtained. On ordinary clean ores the addition of one pound of kerosene oil would produce a raw oil effect on the cells and kill the froth. The only explanation that could be advanced was that the presence of soluble sulphates in the pulp caused the oil to form a stable emulsion. It was found that if the soluble salts were subsequently washed from the pulp, the emulsion was broken down and the oil appeared on the surface of the cells.

When lime is used it should always be added to the ball mill feed, as long contact with the pulp is essential owing to the time required for the lime to go into solution. Lime is used in the selective flotation of copper sulphide ores to prevent the iron sulphides, also the zinc sulphides, if present, from floating. In the selective flotation of lead-zinc sulphide ores, it prevents the zinc sulphide, as well as the iron sulphides, from floating. In the flotation of antimony ores it has been found to absolutely prevent stibnite from floating. It is used in the flotation of molybdenite and graphite, where iron sulphides are present, for the same purpose. The quantity added varies from one to twenty pounds per ton of ore.

Soda ash.—The addition of soda ash to the flotation pulp will increase the flotation of zinc sulphides, copper and nickel sulphides and stibnite, and has a tendency to prevent iron sulphides from floating. The quantity used varies from one to ten pounds per ton of ore.

Sodium sulphite.—This reagent is used in the selective flotation of lead-zinc sulphide ores. It prevents zinc sulphide from floating but has a slight tendency to float iron sulphides, so that if these are present in any quantities, lime must be used also.

Copper sulphate.—The addition of one to two pounds of copper sulphate per ton of ore, when an alkaline pulp is used, will generally assist the flotation of zinc sulphide.

CONTROL OF FLOTATION PULP

The control of the pulp in a flotation plant is very important and should be given as careful attention as in a cyanide plant. Commencing at the discharge of the primary, wet-grinding mills, control samples should be taken of each succeeding step in the operations at various intervals of time. The correct condition of the pulp for flotation must be obtained before it enters the flotation cells. Characteristics displayed by the pulp in the ball mill discharge and in the classifier circuits will in most cases indicate to the operator how the cells are working and whether the pulp is in the right condition for flotation.

Density control.—Careful control must be maintained of the density of flotation pulps. The densities should be kept as uniform as possible and only vary between very narrow limits. The reason for this is obvious, as any variation in the dilution of the pulp will change the concentration of the modifying reagents used to increase the selective flotation. It is often found necessary to sacrifice efficiency in the grinding circuits to obtain efficiency in the flotation circuit. For instance, in order to obtain a low tailing in the selective flotation of a copper-iron sulphide ore, it has been

necessary to maintain a pulp density in the flotation cells of 1:2.5. Dewatering of the flotation feed could not be practised because a definite protective alkalinity with lime must be maintained in the cells to prevent the iron sulphides from floating.

Temperature control.—The control of temperature in the grinding mills is very important in the case of easily oxidized ores, such as the copper-nickel ores of the Sudbury district and copper ores where the chalcopyrite is associated with large quantities of iron sulphides. Trouble in this connexion is seldom met with in conducting the small scale tests. The temperature can be controlled by pulp dilution, but to do so generally means sacrificing the grinding efficiency.

The control of temperatures in the flotation circuits for selective flotation is practised in certain plants where large tonnages are handled to raise the grade of the concentrate being produced. It introduces an extra item of cost and should only be done when absolutely essential, and where a profit can be shown as against flotation at ordinary temperatures.

Control of soluble salts.—In conducting selective flotation tests, the greatest care should be taken to obtain samples which will truly represent the ore to be concentrated. The samples should be of freshly mined ore. In an ore consisting of a large percentage of sulphides, soluble salts form very rapidly. The writer has found that the lead and nickel sulphides in samples of lead and copper-nickel ores, crushed to 10 mesh, will oxidize so rapidly that, in as short a time as ten days, the flotation properties of the minerals will be seriously affected. All ores contain soluble salts which are generally sulphates of iron, aluminium, calcium and magnesium. These salts play a very important part in selective flotation and no definite rules can be given as to how they will act. Large amounts of them will interfere with and even prevent flotation. Small amounts either interfere or improve the selective flotation of one mineral from another. In conducting selective flotation tests, the amount and nature of the salts should be determined for every ore. The subject of overcoming or taking advantage of their effect is too large a one to consider in detail in this article. Further reference is made to them in dealing with the type of testing apparatus.

Control of modifying reagents.—Samples of the water from the flotation cell tailing should be taken at regular intervals and analysed for protective amounts of modifying reagents.

EXAMPLES OF SELECTIVE FLOTATION PROBLEMS

A copper-zinc-iron sulphide ore.—This particular ore was a complex association of copper, zinc and iron sulphides on which grinding to 200 mesh was necessary to free the minerals. It contained chalcopyrite 7 per cent, zinc blende 7 per cent, iron sulphides 73 per cent, and gangue minerals 13 per cent.

It was found that a high-grade copper concentrate could be obtained by using lime to prevent the pyrite and zinc blende from floating. The pulp, after flotation of the chalcopyrite, was dewatered to eliminate the lime. Fresh water and soda ash were added for the flotation of the zinc blende. The governing condition for the separation was in the control of

the lime added for the selective flotation of the chalcopyrite. It was found that lime, in excess of the requirements to produce a high-grade copper concentrate, was necessary to prevent the pyrite from floating with the zinc blende. The addition of lime to the zinc heads did not produce the same effect, nor did running the pulp direct from the copper cell without dewatering. In both cases so much lime was present that the zinc blende could not be floated. The lime had to be added to the grinding circuit before flotation of the chalcopyrite. Investigating the reason for this it was found that, apparently, a skin formed around the grains of pyrite resulting in a greater concentration of lime around the pyrite than in the remainder of the pulp. The concentration of the lime around the grains would vary according to the amount of lime in the pulp, so that it was necessary, in the first place, to add enough lime to obtain a concentration in the film that would maintain the wetting effect on the pyrite after dewatering for the selective flotation of the zinc blende.

A copper-iron sulphide ore.—This was a heavy sulphide ore of copper containing 3 per cent copper as chalcopyrite, 39 per cent iron as pyrite, and 8 per cent insoluble. Grinding to 100 mesh was necessary to free the minerals. An alkaline pulp was maintained by the addition of 10 pounds of lime to the ball mill. The most important point was found to be the control of the pulp density in the flotation cells. A low tailing could only be obtained when a density of 1 : 2.5 to 1 : 3 was maintained in the cells. Under these conditions a concentrate containing 22 per cent copper with a recovery of 95 per cent of the copper values was obtained from this ore without the least difficulty.

Lead-zinc-iron sulphide ores.—The following method has been proved to have the most general application. Lime and sodium sulphite are added and ground in contact with the ore in the ball mills. The lime is added to keep the iron down, and the sodium sulphite to keep the zinc from floating. After the flotation of the lead, the tailing pulp from the lead cells is dewatered, eliminating the bulk of the lime and sodium sulphite. The pulp is then made up to the required density with fresh water, and soda ash and copper sulphate added for the flotation of the zinc. This method was used to obtain the separations given in examples Nos. 1 and 2 in the accompanying tables. The method used by the Consolidated Mining and Smelting Co., at Trail, to concentrate the ore from the Sullivan mine has also been used with success. An ore containing 1.96 per cent lead, 5.52 per cent zinc and 7.42 per cent iron, all as sulphides, was treated by this method. Example No. 3 in the accompanying tables gives the results of a tonnage test on the ore. In this particular case the ore was ground to 65 mesh to free the minerals. The oils used on the Sullivan ore were found to be unsatisfactory, but no change in the addition reagents was necessary. Soda ash at the rate of 5 pounds per ton was fed to the ball mill together with a mixture of coal tar and coal tar creosote. A little cresylic acid was added to the lead cells together with 0.1 pound per ton of sodium cyanide. The tailing from the lead cell went to a mixer where the Barretts No. 634 oil, TT mixture, and 1 pound copper sulphate per ton were added for the flotation of the zinc in the zinc cells.

A zinc-iron sulphide ore.—The selective flotation of the zinc can generally be obtained by the use of an alkaline soda ash pulp, but in some cases the soda ash will not hold down the iron and it is necessary to use

lime. When lime is used it should be added in the ball mills and the pulp dewatered before the soda ash is added. Copper sulphate will often increase the recovery of zinc.

The oils used in the above methods of separation have been purposely omitted. The reason for this is that while the addition reagents have been found to give consistent results on a variety of ores, the oils have not.

EXAMPLES OF SELECTIVE FLOTATION

The following tables contain examples of separations obtained in the laboratory of the Department of Mines by selective flotation:—

Separation of copper-zinc-iron sulphide ore:—

Approximate analysis: Chalcopyrite 7 per cent, zinc blende 7 per cent, iron sulphides 73 per cent, gangue minerals 13 per cent.

Product	Analysis				Distribution of values		
	Cu. per cent	Zn. per cent	Fe. per cent	Au. oz.	Cu. per cent	Zn. per cent	Au. per cent
Copper concentrate.....	16.7	4.4	24.3	0.10	87.9	10.6	18.2
Copper middling.....	1.8	8.7	28.3	0.10	1.89	4.17	3.6
Zinc concentrate.....	0.45	40.7	17.9	0.11	1.27	52.52	10.8
Zinc middling.....	0.5	8.9	33.7	0.07	1.34	10.90	6.6
Tailing.....	0.3	1.9	35.4	0.07	7.57	21.84	60.8

Separation of copper-iron sulphide ore:—

Analysis: Copper 2.98 per cent, Fe. 39.0 per cent, insoluble 8.0 per cent.

Product	Analysis Cu. per cent	Recovery of copper per cent
Copper concentrate.....	20.20	93.3
Tailing.....	0.12	

Separation of lead-zinc-iron ores:—

Analysis: Lead 17.6 per cent, zinc 28.88 per cent, silver 13.45 oz. per ton.

EXAMPLE No. 1

Product	Analysis			Distribution of values		
	Pb. per cent	Zn. per cent	Ag. oz.	Pb.	Zn.	Ag.
Lead concentrate.....	60.0	8.5	42.0	76.2	6.5	69.6
Zinc concentrate.....	7.0	45.9	6.6	21.6	86.5	26.7
Tailing.....	1.6	8.5	2.1	2.1	6.8	3.6

EXAMPLE No. 2

Analysis: Lead 11.40 per cent, zinc 11.40 per cent, iron 32.25 per cent.

Product	Analysis		Distribution of values	
	Pb. per cent	Zn. per cent	Pb.	Zn.
Lead concentrate.....	52.60	6.10	63.0	7.5
Lead middling.....	21.9	10.0	9.4	4.3
Zinc concentrate.....	8.3	37.4	17.1	77.3
Zinc middling.....	9.8	11.1	2.1	2.4
Tailing.....	1.67	1.7	8.4	8.5

EXAMPLE No. 3

Analysis: Lead 1.96 per cent, zinc 5.52 per cent, iron 7.42 per cent, gold 0.02 oz. per ton silver 3.23 oz. per ton.

Product	Analysis						Distribution of values	
	Pb. per cent	Zn. per cent	Cu. per cent	Insol. per cent	Au. oz.	Ag. oz.	Pb.	Zn.
Lead concentrate.....	51.09	4.17	2.60	16.13	0.30	66.20	89.4	94.8
Zinc concentrate.....	1.09	46.19	
Tailing.....	0.15	0.32	

Separation of copper-nickel pyrrhotite ore:—

Analysis: Copper 1.56 per cent, nickel 1.24 per cent.

Product	Analysis			Distribution of values	
	Cu. per cent	Ni. per cent	Cu.+Ni. per cent	Cu.	Ni.
Concentrate.....	12.5	9.6	22.10	97.2	85.6
Tailing.....	0.05	0.20	2.8	14.4

SMALL SCALE TESTING APPARATUS FOR SELECTIVE FLOTATION

The majority of testing laboratories are equipped with the small batch type of laboratory apparatus. The writer has found that the results obtained in this type of machine are, in many cases, unreliable and cannot be obtained in practice. This particularly applies to selective flotation tests and to tests on ores containing appreciable amounts of soluble salts. The batch machine is very satisfactory for preliminary work, but a continuous feed type of small testing unit with its own grinding circuit should be used to check the results. In this latter type of testing apparatus, the reagents can be added continuously with the feed, as in practice, and not all at once which is generally the case in conducting batch tests. The batch method will usually give a better separation than can be obtained

in practice. If a small scale flotation machine, with a continuous grinding circuit, is not available, the following procedure should be tried out to check the batch methods of adding reagents. During the grinding of the sample the ball mill should be stopped, opened up, and portions of the total amount of reagents to be used added from time to time instead of all at once, and in this way conditions more nearly approaching mill practice are obtained. In conducting flotation tests, the ore should be prepared by wet grinding. Dry ore should never be fed to the flotation machine unless dry crushing is to be adopted in the milling practice.

CONCLUSIONS

The application of selective flotation has increased rapidly in recent years. Although it is a new field in ore-dressing, the scope of its usefulness is gradually being enlarged. It will play a very important part in the economic treatment of complex ores and will permit the commercial development of many ore-bodies which have hitherto been thought unprofitable and will prove to be an important factor in the development of our mineral resources.

The purpose of this paper is to introduce the subject of selective flotation in the hope that other investigators will publish the results of their work and the methods used both in the laboratory and in practice. As yet, very little information has been published. It would be to the mutual advantage and benefit of all concerned if it were more openly and freely discussed.

V

EXPERIMENTAL TESTS ON THE BENEFICIATION OF
CANADIAN IRON ORES

W. B. Timm

The Ore Dressing and Metallurgical Laboratories of the Mines Branch, Department of Mines, Ottawa, were first equipped in 1909 for experimental work on iron ores, and previous to the war a large amount of test work was conducted on the beneficiation of Canadian iron ores. During the war the laboratories were used almost entirely for work on war minerals, and since the war for experimental test and research on non-ferrous ores. The progress made on the beneficiation of iron ores has been closely watched, and the laboratory equipment for this work kept up to date by the purchase of improved apparatus, so that the pre-war work can be supplemented by further investigations, when, by prospecting and development work, representative ore is made available for test.

Experimental work has been conducted on shipments of a few hundred pounds to forty-five tons, from various localities, such as the Bristol mines, Bristol, Que., the Bathurst mine, Bathurst, N.B., the Wilbur mine, Wilbur, Ont., the Culham mine, Lanark, Ont., Nictaux-Torbrook mine, Torbrook, N.S., Goulais river, Algoma district, Ont., Natashkwan river, lower St. Lawrence, Que., the Bessemer mine, Bessemer, Ont., the Childs mine, Bessemer, Ont., Riviere des Rapids, Saguenay, Que., Groundhog river, Sudbury dist., Ont., Lavant mine, Lanark, Ont., Kaministiquia, Ont., and Clarendon, Ont. Results of these tests have been fully described in the Summary Reports of the Mines Branch.

The following is a brief summary of the more-important results of the experimental tests on the majority of the above ores:—

Wilbur Mine Ore—Shipment 10 Tons

Location of property.—Lots 3 and 4, con. IX, XII, XIII, Lavant tp., Lanark co., Ont.

Character of ore.—Moderately fine crystalline magnetite, the gangue consisting for the most part of quartz, calcite, and chlorite in seams and stringers throughout the ore. Other gangue minerals present in smaller amounts are hornblende and muscovite. Both sulphur and phosphorus are low.

Results of preliminary cobbing tests.—These tests showed the magnetite fairly free at about 40 mesh and a 60 per cent iron concentrate with a low tailing was made at this mesh.

Results of wet magnetic separation tests.—Screen analysis of ball mill discharge was—on 100 mesh, 24 per cent; on 200 mesh, 67 per cent;

through 200 mesh, 33 per cent. Analyses of crude ore, concentrate, and tailing, were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	48.5	66.10	7.2
Insoluble residue.....	13.08	3.30	
Sulphur.....	0.105	0.028	
Phosphorus.....	0.011	0.004	
Lime.....	3.00	0.30	
Magnesia.....	6.40	1.40	

Ratio of concentration..... 1 : 1.43
 Recovery..... 95.3 per cent

Wilbur Mine Waste Dump—Shipment 5 Tons

Character of ore.—Similar to run-of-mine. Major portion of gangue granitic gneiss; calcite and chlorite present in about the same proportion as in run-of-mine.

Results of wet magnetic separation tests.—Screen analysis of ball mill discharge was—on 100 mesh, 32 per cent; on 200 mesh, 63.5 per cent; through 200 mesh, 36.5 per cent. Analyses of crude ore, concentrate, and tailing, were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	38.2	64.6	5.0
Insoluble residue.....	22.18	6.11	
Sulphur.....	0.101	0.023	
Phosphorus.....	0.011	0.006	
Lime.....	3.20	0.14	
Magnesia.....	7.0	1.64	

Ratio of concentration..... 1 : 1.8
 Recovery..... 94.5 per cent

General conclusions.—A high-grade iron concentrate can be made from both the run-of-mine and the lower grade waste ore at fairly coarse grinding. The concentrate would have to be sintered, but the resultant product would be very satisfactory for blast furnace use.

Robertsville Mine Waste Dump—Shipment 5 Tons

Location of property.—Lots 3 and 4, con. IX, Palmerston tp., Frontenac co., Ont.

Character of ore.—Fairly coarse crystalline magnetite in a gangue of diorite country rock. Black hornblende and pink calcite with smaller amounts of white quartz is also associated with the magnetite. Sulphur and phosphorus present in small amounts.

Results of wet magnetic separation tests.—Analyses of crude ore, concentrate and tailing:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	28.00	70.5	3.6
Insoluble residue.....	50.70	3.1	
Sulphur.....	0.004	0.00	
Phosphorus.....	0.028	0.014	
Lime.....	3.30	0.00	
Magnesia.....	1.90	0.06	

Ratio of concentration..... 1 : 2.74

Recovery..... 91.8 per cent

General conclusions.—A high-grade iron concentrate was made with a high recovery. The concentrate would have to be sintered, but the resultant product would be very satisfactory for blast furnace use.

Culham Mine Ore—Shipment 3 Tons

Location of property.—N. $\frac{1}{2}$ lot 21, con. VII, Bagot tp., Renfrew co., Ont.

Character of ore.—Moderately fine-grained crystalline magnetite, the gangue being schistose material, calcite and iron pyrites. Both sulphur and phosphorus are present in objectionable amounts.

Results of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	47.7	67.0	6.6
Insoluble residue.....	9.3	2.5	
Sulphur.....	1.65	0.357	
Phosphorus.....	0.179	0.074	
Lime.....	4.20	0.10	
Magnesia.....	0.66	0.05	

Ratio of concentration..... 1 : 1.47

Recovery..... 95.5 per cent

General conclusions.—A high-grade iron concentrate was made with a high recovery. The phosphorus content is above the Bessemer limit, but low enough for mixing purposes. Sintering would be required, and this would reduce the sulphur content and make the resultant product satisfactory for blast furnace use.

Bathurst Mine Ore—Shipments 15 and 10 Tons

Location of property.—Lot 12, range XVII, Bathurst tp., Gloucester co., N.B.

Character of ore.—A compact cryptocrystalline mixture of hematite, magnetite and quartz. The ore possesses a slightly laminated structure, although an alternate banding of hematite, magnetite and quartz is not

apparent to the naked eye. Hematite and magnetite appear to be intimately associated in about equal proportions, the colour and streak of the ore varying from red to black, according to the proportion of ferric iron.

Results of wet magnetic separation tests.—Analyses of crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	48.7	60.3	38.2
Ferrous oxide.....	18.1	23.8	9.6
Ferric oxide.....	49.4	59.7	43.8
Insoluble residue.....	16.9	7.7	
Sulphur.....	0.127	0.046	
Phosphorus.....	0.760	0.350	

Ratio of concentration..... 1 : 2.1
 Recovery..... 58.8 per cent

General conclusions.—A good grade of concentrate was produced, but with a low recovery of the iron. The loss in iron was due to the hematite in the crude reporting in the tailing. This method of concentration is not adaptable to the Bathurst ore. Magnetic cobbing tests and jig tests were made on carefully sized ore, but by both these methods it was found that the grade could not be raised any appreciable extent without a high loss of the iron content. It would seem that concentration methods beyond a rough cobbing to remove any waste material broken with the ore are not applicable to Bathurst ore.

Torbrook Mine Ore, "Hematite Vein"—Shipment 15 Tons

Location of property.—Nictaux-Torbrook area, Annapolis co., N.S.

Character of ore.—The Torbrook hematite vein is composed of hard compact ore of finely crystalline magnetite and hematite of the specular variety. The gangue is predominantly siliceous, although lime and magnesia are present in small amounts. Hematite and magnetite exist in about the proportion of 1 to 2.6 respectively.

Results of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	47.2	61.0	33.0
Ferrous oxide.....	14.9	21.6	9.4
Ferric oxide.....	50.8	63.7	36.7
Insoluble residue.....	21.7	9.9	
Sulphur.....	0.011	0.005	
Phosphorus.....	1.41	0.54	
Magnesia.....	0.86	0.28	

Ratio of concentration..... 1 : 1.97
 Recovery..... 65.6 per cent

Torbrook Mine Ore, "Shell Vein"—Shipment 15 Tons

Character of ore.—Very similar in physical characteristics to the hematite vein, although the former is more magnetic. The ore is siliceous, but contains considerably more lime than the hematite vein.

Results of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	42.5	60.5	22.8
Ferrous oxide.....	19.0	21.8	8.1
Ferric oxide.....	39.6	62.1	23.6
Insoluble residue.....	18.4	8.5	
Sulphur.....	0.033	0.005	
Phosphorus.....	1.03	0.62	
Lime.....	5.50	0.58	
Magnesia.....	0.37	trace	

Ratio of concentration..... 1 : 1.91

Recovery..... 74.5 per cent

General conclusions.—The Torbrook ores being a mixture of magnetite and hematite, the greater proportion of the hematite is lost in the tailing by fine grinding and magnetic concentration. A concentrate over 60 per cent iron was made in both cases, with low sulphur content but with phosphorus over one-half per cent. Recoveries of 65 per cent to 75 per cent were made depending on the proportion of magnetite in the ores. Hematite slimes badly and it is, therefore, doubtful if tabling of the tailings from magnetic concentration would pay.

Goulais River Ore—Shipment 15 Tons

Location of property.—Tp. 22, range XII, Algoma district, Ont.

Character of ore.—The ore is extremely fine-grained. It consists almost entirely of a mixture of silica and magnetite alternating in narrow bands, the two constituents varying widely in their proportions. The leaner bands of quartz contain considerable iron and the richer bands of magnetite appreciable amounts of silica. The average iron content of the shipment was about 35 per cent. Sulphur is low, being about 0.05 per cent.

Result of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	33.9	50.5	4.3
Insoluble residue.....	52.1	30.9	
Sulphur.....	0.056	traces	
Phosphorus.....	0.090	0.046	
Lime.....	0.20		
Magnesia.....	0.10		

Ratio of concentration..... 1 : 1.57

Recovery..... 94.8 per cent

A screen analysis of the concentrate shows 35 per cent on 200 mesh, 65 per cent through 200 mesh. The -200-mesh concentrate contained 61.4 per cent iron, showing that final grinding to 200 mesh is necessary to produce a product over 60 per cent iron.

General conclusions.—From the nature of the ore it would seem that the Babbitt practice of stage crushing, sizing and cobbing, for the elimination of waste and final reduction and separation at 200 mesh would be applicable, resulting in a product, on being sintered, better than 60 per cent iron, low sulphur and phosphorus within the Bessemer limits, highly desirable for blast furnace use.

Natashkwan Sands—Shipment 45 Tons

Location of property.—Natashkwan river, lower St. Lawrence, Saguenay district, Que.

The shipments made consisted of the samples obtained from an accurate survey and sampling of the deposits. The sands occur at the mouth of the Natashkwan river on the north side of the lower St. Lawrence, for a distance of 6 miles east of the present mouth of the river. The sands consist of rounded particles of titaniferous magnetite and silica, washed down by the river and deposited at its mouth. Concentration of the sands has taken place by wave and wind action along the coast.

Results of wet magnetic separation tests.—Average analyses of crude ore, 1st and 2nd concentrates, and tailings:—

—	Crude ore	1st concentrate	2nd concentrate	Tailing
	per cent	per cent	per cent	per cent
Iron.....	9.60	64.61	68.37	5.61
Titanic oxide.....	2.26	2.36	1.61	2.69
Silica.....		6.35	2.27	84.58
Sulphur.....			trace	
Phosphorus.....			0.023	
Manganese.....			0.19	
Lime.....			0.13	
Magnesia.....			0.145	

Ratio of 1st concentration	1 : 14.8	Recovery.....	45.5 per cent
" 2nd "	1 : 1.06	" 	96.0 "
Final ratio of	1 : 16.2	Final recovery.....	43.7 "

NOTE.—Iron analysis given above is total iron, and not magnetic iron, hence the low recovery by magnetic separation.

General conclusions.—The magnetic iron sands of the lower St. Lawrence can be concentrated up to a high-grade iron product, low in sulphur and phosphorus, but containing up to 2 per cent titanic oxide. The concentrate would require sintering for blast furnace use.

Bessemer Mine Ore—Shipment 2 Tons

Location of property.—Lot 4, con. VI, Mayo tp., Hastings co., Ont.

Character of ore.—Fine-grained magnetite, with an associated gangue of calcite, garnet, epidote and other silicates. The magnetite and gangue are, however, intimately mixed, and it is possible to cob out a large proportion of the gangue before final separation.

Result of wet magnetic separation tests.—Analyses of crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	36.5	67.4	4.5
Insoluble residue.....	35.37	5.87	
Sulphur.....	0.314	0.185	
Phosphorus.....	0.026	0.007	
Lime.....	5.68		
Magnesia.....	0.30		

Ratio of concentration..... 1 : 1.96

Recovery..... 94.0 per cent

A screen analysis of the ball mill discharge shows:—On 100 mesh, 22 per cent; on 200 mesh, 50 per cent; through 200 mesh, 50 per cent.

General conclusions.—The screen analysis shows that very fine grinding is not necessary to produce a high grade concentrate. The concentrate would require sintering, but the resultant product would be high grade, low in sulphur and phosphorus, and highly desirable for blast furnace use.

Bessemer Mine Ore—Shipment 5 Tons

Results of wet magnetic separation, and sintering tests.—Analyses of crude ore, concentrate and sinter were:—

	Crude ore	Concentrate	Sinter
	per cent	per cent	per cent
Iron.....	43.9	67.5	66.2
Insoluble.....	30.83	5.98	
Sulphur.....	0.3	0.11	0.023
Phosphorus.....	0.028	0.019	0.016

The sinter produced was hard and extremely porous, an ideal product for blast furnace use.

Childs Mine—Shipment 2 Tons

Location of property.—Lots 11 and 12, con. IX, Mayo tp., Hastings co., Ont.

Character of ore.—Granular magnetite with an associated gangue of calcite, pyroxene, chlorite, etc. Although the ore is quite soft, the gangue minerals are so intimately associated with the magnetite that fine grinding is necessary to effect a satisfactory separation of the magnetite from the gangue.

Results of wet magnetic separation tests.—Analyses of crude ore, concentrate, and tailing were:—

—	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron.....	35.0	66.4	5.7
Insoluble residue.....	36.8	6.09	
Sulphur.....	0.045	0.022	
Phosphorus.....	0.083	0.016	
Lime.....	5.83		
Magnesia.....	0.41		

Ratio of concentration..... 1 : 2
 Recovery..... 92 per cent

A screen analysis of the ball mill discharge showed:—On 100 mesh, 33 per cent; on 200 mesh, 63 per cent; through 200 mesh, 37 per cent.

General conclusions.—The screen analysis shows that very fine grinding is not necessary to produce a high-grade concentrate. The concentrate would require sintering but the resultant product would be high grade, very low in sulphur and phosphorus, and very desirable for blast furnace use.

Groundhog Ore—Shipment 15 Tons

Location of property.—Groundhog river, tps. of Keith and Penhorwood, Sudbury district, Ont.

Character of ore.—Banded structure, consisting of bands of magnetite, hematite and jasper, the bands ranging from one-half inch in width down to a very small fraction of an inch. The magnetite occurs almost pure in some cases, in others mixed with the jasper, and in other cases finely disseminated throughout the jasper. Coarse crushing will not free the thinner bands of magnetite nor the hematite, as the hematite is in most cases firmly attached to the jasper. The shipment showed an average analysis of 35 per cent iron, of which 26.5 per cent was magnetite, and 8.5 per cent hematite, with only traces of sulphur, phosphorus and other impurities. The problem was therefore simply one of raising the iron content.

Results of experimental tests.—A large number of tests were conducted, including magnetic cobbing, jig and table concentration, and wet magnetic separation tests. Magnetic cobbing tests were made on the sized material from 1 inch down. These tests showed that no appreciable amount of concentrate approaching 55 per cent metallic iron could be recovered in sizes coarser than 1/16 inch. By crushing to 1/16 inch, magnetic cobbing, jig and table concentration of the sized 1/16-inch tailings, a concentrate was produced, assaying 53.5 per cent iron, with a recovery of 73.2 per cent and with a ratio of concentration 1:2.

Wet magnetic separation tests on this ore, followed by tabling of the tailings, after grinding to 69 per cent through 200 mesh, gave a magnetic product assaying 64 per cent iron; and a hematite product assaying from 50 per cent to 55 per cent iron, with a recovery of total iron up to 85 per cent. The ratio of concentration was 1:1.8.

General conclusions.—The results of the experimental tests show that a high recovery of the iron content cannot be expected on this class of ore by any simple method of concentration. A fair grade of concentrate was made after grinding to 1/16 inch, and a much better grade on finer grinding. The concentrate would require sintering for blast furnace use. The sinter would be high in silica and low in sulphur and phosphorus.

Kaministiquia Ore—Shipment 240 Pounds

Location of property.—Marks-Wiley property, Loon lake, Thunder Bay district, Ont.

Character of ore.—Banded structure, very similar to Groundhog ore.

Results of experimental tests.—Crushing to 4 mesh, jig and table concentration of the sized material gave:—

Analysis of crude ore:—	Total iron (Fe.).....	37.19 per cent
	Hematite (Fe_2O_3).....	17.98 “
	Magnetite (Fe_3O_4).....	33.97 “
Analysis of concentrate: Iron.....		55 “
Analysis of tailing: Iron.....		20 “
Recovery of iron content.....		70 to 75 “
Ratio of concentration.....		1 : 2

General conclusions.—Concentrate too fine for blast furnace use. Would require sintering. Product high in silica, low in sulphur and phosphorus.

The equipment of the Mines Branch laboratories consists of small and large scale apparatus and machinery permitting of experimental tests on a few hundred pounds, and on carload lots. With the present equipment, a flow sheet duplicating, on a small scale, the Babbitt practice can be followed, and various combinations can be set up to permit of deviations therefrom. Roasting and sintering tests can be made on a small scale. With very little additional equipment the laboratory investigations could be extended to the point where the details of operation might be worked out in a pilot plant. The chemical laboratories of the Division are equipped for making all necessary determinations in connexion with the experimental work by means of the Davis testing tube.

The small scale equipment of the laboratories consists of jaw crusher, rolls, pulverizer, screen frame for sizing, jigs, Wilfley table, classifiers, magnetic separators, rotating kiln, roasting furnace, and a Davis magnetic testing machine for the determination of magnetic iron.

The larger scale equipment consists of Blake crusher, picking belt, two sets of rolls, Hardinge ball mill, Traylor ball mill, automatic samplers, two sets of jigs, standard size Wilfley table, Ferraris screen, Hammer screen, Callow duplex screen, Dorr simplex classifier with bowl attachment, launder classifier, Grondal magnetic cobber, Grondal magnetic separator, Davis magnetic log washer, Ullrich magnetic separator, roasting furnaces, duplex Dwight Lloyd sintering pans.

VI

THE SELECTIVE FLOTATION OF THE LOWER GRADE
NICKELIFEROUS PYRRHOTITE ORES OF ONTARIO

W. B. Timm

INTRODUCTORY

The practice of smelting the copper-nickel ores of the Sudbury district direct in the blast furnace, or by mixing the green ore with a certain portion of roasted product, or by smelting the roasted fines in a reverberatory furnace, has given very favourable results. This practice has been especially adaptable to the higher grade ores when consideration is given to the favourable market conditions that have prevailed to within the last few years.

In the treatment of the lower grade ores of the district, of which there are enormous reserves, it is a question whether it would not be more economical to apply certain methods of concentration to produce a product with a fairly high copper-nickel content prior to smelting operations. One of the operating companies has for several years been treating a portion of their lower grade material by concentration on tables and by flotation. By sintering these concentrates with flue dust and mine fines they obtain a roasted product very desirable in many ways for mixing with green ore for the blast furnace charge.

Experimental tests were conducted in the Mines Branch laboratories at Ottawa to determine whether a large portion of the barren pyrrhotite, as well as the siliceous gangue, could be eliminated, with the production of a concentrate having a copper-nickel content similar to that of the matte now being produced. Further, as the success of any such process would depend to a large extent on the recovery of the precious metal values, as well as the copper-nickel values, the experimental work included a study of the concentration products, to determine whether the precious metal values were reporting in the copper-nickel concentrate.

THE SELECTIVE FLOTATION OF THE LOWER GRADE COPPER-NICKEL
ORES OF THE SUDBURY DISTRICT, ONTARIO

Experimental tests.—The first series of tests was made on the lower grade ores from two mines, to determine whether a high-grade concentrate could be made by selective flotation with high recoveries of the copper and nickel values, by the elimination of the gangue minerals and a large portion of the barren pyrrhotite. No attention was paid to where the precious metal values were reporting in this series of tests.

Table I gives the results of tests Nos. 1, 2, 4 and 5, on an ore of the following analysis:—

Copper.....	1.25 per cent
Nickel.....	1.35 "
Iron.....	23.70 "
Sulphur.....	10.70 "
Silica.....	37.85 "

Recoveries indicated include the actual recovery made in the concentrate plus 50 per cent of the values in the middling that would be recovered by returning this product to the head of the circuit. This percentage was proven to be a conservative estimate.

Table II gives the results of tests Nos. 6 and 8 on the same ore. The middling was re-run without further grinding. By regrinding the middling product, higher recoveries could be expected.

Table III gives the results of tests Nos. 1, 2, 3, 4, 5, 6 and 8 on a still lower grade ore, of the following analysis:—

Copper.....	0.47 per cent
Nickel.....	1.17 “
Iron.....	20.90 “
Sulphur.....	11.19 “
Silica.....	42.15 “

TABLE I

Test No.	Concentration products	Weight grms.	Analysis			Percentage of values		Total recoveries		
			Cu. per cent	Ni. per cent	Cu. +Ni. per cent	Cu.	Ni.	Cu. per cent	Ni. per cent	Cu. +Ni. per cent
1	Concentrate	106	12.50	9.60	22.10	93.2	76.2	95.3	82.4	89.1
	Middling...	147	0.40	1.13	4.1	12.4			
	Tailing.....	758	0.05	0.20	2.7	11.3			
2	Concentrate	75	16.50	10.20	26.70	88.3	58.7	91.9	68.8	80.8
	Middling...	155	0.65	1.70	7.2	20.2			
	Tailing.....	783	0.08	0.35	4.5	21.1			
4	Concentrate	95	13.66	9.75	23.41	93.6	69.8	95.4	78.3	87.1
	Middling...	146	0.35	1.55	3.7	17.0			
	Tailing.....	758	0.05	0.23	2.7	13.1			
5	Concentrate	170	7.40	6.20	13.60	92.0	80.0	93.9	84.5	89.2
	Middling...	94	0.55	1.27	3.8	9.0			
	Tailing.....	730	0.08	0.20	4.2	11.0			

TABLE II

Test No.	Concentration products	Weight grms.	Analysis			Percentage of values		
			Cu. per cent	Ni. per cent	Cu. +Ni. per cent	Cu.	Ni.	Cu. +Ni.
6	Concentrate.....	376	7.60	5.70	13.30	91.4	86.5	89.2
	Tailing.....	1,594	0.17	0.21	0.38	8.6	13.5	10.8
8	Concentrate.....	292	9.10	7.50	16.60	96.7	83.4	90.2
	Tailing.....	1,675	0.055	0.26	0.31	3.3	16.6	9.8

TABLE III

Test No.	Concentration products	Weight grms.	Analysis			Percentage of values			Total recoveries		
			Cu. per cent	Ni. per cent	Cu. + Ni. per cent	Cu.	Ni.	Cu. + Ni.	Cu. per cent	Ni. per cent	Cu. + Ni. per cent
1	Concentrate.....	155	2.6	6.2	8.8	87.4	82.6	84.0	90.0	86.8	87.8
	Middling.....	159	0.15	0.62	5.2	8.4				
	Tailing.....	694	0.05	0.15	7.4	8.9				
2	Concentrate.....	175	2.30	5.75	8.05	89.3	84.7	86.0	90.7	87.0	88.0
	Middling.....	118	0.10	0.45	2.7	4.5				
	Tailing.....	715	0.05	0.18	8.0	10.8				
3	Concentrate.....	194	2.15	5.26	7.41	89.7	87.1	82.4	90.9	81.6	84.1
	Middling.....	82	0.15	0.57	2.6	3.6				
	Tailing.....	730	0.05	0.29	7.7	10.6				
4	Concentrate.....	220	1.90	4.80	6.70	91.5	88.5	89.3	92.3	90.3	90.8
	Middling.....	144	0.05	0.30	1.5	3.6				
	Tailing.....	630	0.05	0.15	7.0	7.9				
5	Concentrate.....	64	3.05	7.12	10.17	83.7	76.6	78.6	86.3	80.0	81.8
	Middling.....	69	0.17	0.56	5.1	6.5				
	Tailing.....	369	0.07	0.27	11.2	16.8				
6	Concentrate.....	66	6.65	13.36	20.01	82.1	73.0	75.7	84.9	79.6	81.2
	Middling.....	122	0.25	1.33	5.6	13.4				
	Tailing.....	825	0.08	0.20	12.3	13.6				
8	Concentrate.....	168	2.70	5.83	8.53	93.2	80.4	84.0	94.6	85.5	88.1
	Middling.....	205	0.07	0.61	2.9	10.3				
	Tailing.....	632	0.03	0.18	3.9	9.3				

The procedure followed in conducting the tests was:—

The ore was crushed to 20 mesh; 1,000 grammes was ground wet in a small ball mill with the reagents for about 40 minutes; a rougher concentrate and tailing were first made; the rougher concentrate was re-cleaned, making a concentrate and a middling. In tests Nos. 6 and 8 of Table II, two 1,000-gramme lots were used in order to obtain sufficient middling for refloating. The concentrate from refloating the middling was added to the final concentrate, and the tailing to the rougher tailing, so as to have only two products, concentrate and tailing.

No test work was done to determine the degree of fineness to which the ores should be ground to obtain the best results. For the tests, approximately 90 per cent would pass a 200-mesh screen. For test No. 8 of Table III, it was ground to pass a 100-mesh screen, 67 per cent through 200 mesh.

Conclusions from experimental tests.—There is no apparent difficulty in making a fairly high-grade concentrate, with a good recovery of the copper-nickel values. A product was made, in certain cases, with a higher copper-nickel content than that contained in the blast furnace mattes and certainly higher than the mattes that would be produced by present smelting practice on the same grades of ores.

For the selective flotation of the ores, the alphabetical reagents were found to be the most suitable. In using these reagents the essential point is to maintain an alkaline pulp by the addition of lime, soda ash or caustic soda. The chalcopyrite floats with remarkable ease, but the nickel minerals are more difficult to float.

Examination for the amounts of pyrrhotite being eliminated in the tailing was made on the flotation products of the tests run for the determination of precious metal values as given in Tables IV and VI, where the concentration was not nearly so good as in some of the tests given in Tables I and III. This was found to be 30 per cent and 41 per cent respectively, of the amounts in the ore. This elimination of practically barren pyrrhotite, together with the gangue minerals, means considerable savings in several directions. Taking the flotation products of tests Nos. 1, 2 and 4 of Table I and of test No. 6 of Table III, where high-grade concentrates were made, the percentage of pyrrhotite eliminated in the tailings would be much higher than that given above.

THE PRECIOUS METAL VALUES IN THE FLOTATION PRODUCTS OF THE LOWER GRADE COPPER-NICKEL ORES OF THE SUDBURY DISTRICT

Experimental tests.—To determine if the precious metal values in the ores were being concentrated with the copper-nickel values, flotation tests in one kilogram lots were conducted on 13 kilograms of each ore. The products from the thirteen tests on each ore were combined to obtain sufficient quantities of the flotation products for accurate determination of the precious metal contents. In conducting these tests attention was not paid to obtaining the best flotation results, as this was demonstrated by former tests, the results of which are given in Tables I, II and III.

Tables IV and V give the results on the first ore, the flotation results on which are given in more detail in Tables I and II.

Tables VI and VII give the results on the second ore, which was of still lower grade, the flotation results on which are given in more detail in Table III.

TABLE IV

Product	Weight grms.	Assays						
		Nickel per cent	Copper per cent	Gold oz. per ton	Plat. oz. per ton	Palladm. oz. per ton	Rhod., Irid., etc. oz. per ton	Precious metal values content
Concentrate.....	2,050	5.88	7.55	0.036	0.062	0.072	0.006	\$13 18
Middling.....	1,490	0.93	0.20	0.012	0.019	0.022	0.004	4 47
Tailing.....	9,460	0.16	0.05	0.002	0.0027	0.0038	trace	0 62
Ore.....	13,000	1.15	1.25	0.008	0.014	0.017	0.0014	3 04

*Results are no doubt low due to oxidation and volatilization in fusing and scorifying.

**Figures from December, 1922, quotations on these metals.

TABLE V

Product	Percentages in products						Precious metal values
	Weight	Nickel	Copper	Gold	Plat.	Palladm.	
Concentrate.....	15.8	80.6	95.5	66.8	70.3	68.5	68.4
Middling.....	11.5	9.3	1.8	16.1	15.7	16.3	16.8
Tailing.....	72.7	10.1	2.9	17.1	14.0	15.2	14.8

TABLE VI

Product	Weight grms.	Assays						Precious metal values content
		Copper per cent	Nickel per cent	Gold oz. per ton	Plat. oz. per ton	Palladm. oz. per ton	Rhod., Irid., etc. oz. per ton	
Concentrate.....	1,635	2.90	6.40	0.027	0.032	0.043	*	**
Middling.....	1,418	0.27	1.65	0.013	0.016	0.015	0.003	\$7 40
Tailing.....	9,047	0.05	0.21	0.0009	0.0011	0.0026	3 54
	13,000	0.43	1.14	0.0055	0.0067	0.0009	0.0009	0 30
								1 55

*Results are no doubt low due to oxidation and volatilization in fusing and scorifying.

**Figures from December, 1922, quotations on these metals.

TABLE VII

Product	Percentages in products						Precious metal values
	Weight	Copper	Nickel	Gold	Plat.	Palladm.	
Concentrate.....	12.6	84.4	70.3	61.8	60.9	60.0	60.3
Middling.....	10.9	6.8	15.7	25.7	26.4	18.0	24.9
Tailing.....	76.5	8.8	14.0	12.5	12.7	22.0	14.8

Conclusions from examination of flotation products.—The results show, in the case of the first ore, that 68 per cent of the total precious metal values is contained in the concentrate, 17 per cent in the middling product and 15 per cent in the tailing. By regrinding and floating the middling, which represents 11.5 per cent of the original feed, a total recovery of 80 per cent of the precious metal values would be obtained in the copper-nickel concentrate by selective flotation.

The results show, in the case of the second ore of lower grade, that 60 per cent of the precious metal values is contained in the concentrate, 25 per cent in the middling product, and 15 per cent in the tailing. By regrinding and floating the middling, which represents 10.9 per cent of

the original feed, a total recovery of 75 per cent of the precious metal values would be obtained in the copper-nickel concentrate by selective flotation.

The results of the above tests show that the gold and the platinum group metals are reporting to a large extent in the concentrate with the copper and nickel minerals.

In the determination of the precious metal contents, the total flotation products were used, and the assay results obtained figured to the per ton basis. The assays so obtained should be reliable and accurate.

No definite conclusions were arrived at with respect to the particular mineral or minerals with which the platinum group metals are associated. In studying the ratios between these metals and the copper, nickel, iron and sulphur content in the flotation products, there is no definite uniformity of values or proportions between these metals and the chalcopyrite, pentlandite, pyrrhotite, content in the ores and flotation products. Whether this should be accepted in support of the opinion that the lower grade ores carry relatively higher values in the platinum group metals is questionable.

THE SELECTIVE FLOTATION OF THE COPPER-NICKEL ORES OF SHEBAN-DOWAN LAKE DISTRICT, ONTARIO

Experimental tests.—Tests were conducted on a shipment of ore from this district to determine whether a high-grade copper-nickel concentrate could be obtained with a satisfactory recovery of the copper-nickel and precious metal values. The tests were made on an ore of the following analysis:—

Nickel.....	2.97 per cent	Lime	2.50 per cent
Copper.....	1.65 "	Magnesia.....	7.64 "
Cobalt.....	0.15 "	Sulphur.....	15.84 "
Iron.....	24.30 "	Gold.....	0.01 ozs. per ton
Alumina.....	10.15 "	Platinum.....	0.03 "
Silica.....	26.55 "	Palladium.....	0.048 "

Table VIII gives the results of the tests. In test No. 1 a concentrate and tailing were made. In test No. 2 a concentrate, middling and tailing were made.

TABLE VIII

Test No.	Product	Weight per cent	Analysis			Percentage of values			Total recoveries		
			Cu. per cent	Ni. + Co. per cent	Cu. + Ni. + Co. per cent	Cu. per cent.	Ni. + Co.	Cu. + Ni. + Co.	Cu. per cent	Ni. + Co. per cent	Cu. + Ni. + Co. per cent
1	Concentrate.....	25.7	6.35	9.88	16.23	83.0	92.1	88.3	83.0	92.1	88.3
	Tailing.....	74.3	0.45	0.24	17.0	7.9
2	Concentrate.....	30.7	5.00	8.56	13.56	88.3	95.5	92.7	91.1	96.4	94.4
	Middling.....	17.0	0.58	0.30	5.7	1.9			
	Tailing.....	52.3	0.20	0.14	6.0	2.6			

Conclusions from experimental tests.—A high recovery of the copper, nickel and cobalt values was made in a good grade of concentrate. The recovery of the copper is not as good as on the Sudbury ores, probably due to oxidation. Continuous grinding and flotation tests will be made, and the precious metal values in the flotation products determined, to prove whether these values are reporting in the concentrate. Taking the results obtained in test No. 2 of Table VIII, determinations made, show that 36 per cent of the pyrrhotite in the ore is eliminated in the tailing, and this elimination of pyrrhotite, if practically barren as is the case in the tests on the Sudbury ores, together with the gangue silicates, is very desirable for subsequent smelting operations.

VII

DESCRIPTION OF ORE CONCENTRATION PLANTS
IN CANADA

THE NEW SULLIVAN CONCENTRATOR AT KIMBERLEY, B.C.

On August 24, 1923, the Consolidated Mining and Smelting Co. of Canada put into operation its new concentrator at Kimberley, B.C., for the treatment of the Sullivan mine ore. It is the result of years of experimentation in the laboratory and in pilot plants on the treatment of this complex sulphide ore.

The Sullivan ore is a valuable mixture of zinc, lead and iron sulphides with relatively small amounts of gangue, principally calcite and quartz. The zinc content varies from 12 to 15 per cent; the lead content from 10 to 12 per cent; the iron from 32 to 35 per cent; the sulphur from 30 to 33 per cent; the insoluble from 4 to 6 per cent; the silver is about 3 ozs. per ton. The chief minerals are marmatite, galena, pyrrhotite, pyrite, calcite and quartz. A polished section indicates the intimate association of the minerals and fine grinding to at least 200 mesh is necessary to free them.

The concentrator is favourably situated on the slope of a hill, $22\frac{1}{2}^{\circ}$ to the horizontal, and advantage is taken of gravity for the flow of the ore and pulp. Below the concentrator is a large area of 40 acres for tailing disposal. The concentrator is connected with the coarse crushing plant, situated below the portal of the lower adit, by a standard gauge railway, $3\frac{1}{2}$ miles long, and is also connected with the main railway for the shipping of concentrate to the company's smelter at Trail, B.C.

Coarse crushing plant.—In this plant the ore from the mine is reduced to 3 inches in a 36 by 42-inch type C Buchanan jaw crusher and in two No. 8 Gates gyratory crushers, and conveyed to one 2,500-ton railway bin.

Roll plant.—This plant is situated to one side and above the concentrator. It receives its feed from a 1,000-ton ore bin. The ore is here reduced to three-fourths inch in two sets of 72 by 20-inch Alaska type rolls.

Sampling plant.—From the roll plant the ore is conveyed to a 2,500-ton fine ore bin, directly above the concentrator. In passing to this bin a one ten-thousandth portion of the feed is cut out by a system of automatic sampling and mixing. Flood automatic samplers are also used for control samples at various points in the mill.

Fine grinding plant.—In this plant the ore is reduced to the required fineness for flotation, namely 95 per cent through 200 mesh. To obtain this reduction, two 8-foot by 48-inch Hardinge ball mills are used for primary grinding and four 8-foot by 48-inch Hardinge ball mills for secondary grinding. Ten 6 by 25-foot rake Dorr classifiers are used for primary classification and two 10-foot bowl, 3-foot rake Dorr classifiers for secondary classification.

Lead recovery plant.—For the recovery of the lead sulphide, three primary 18-cell, 24-inch modified Minerals Separation flotation machines are used, followed by three steps of cleaner cells each consisting of 8-cell 24-inch modified Minerals Separation flotation machines.

Zinc recovery plant.—For the recovery of the zinc sulphide, four primary 18-cell, 24-inch modified Minerals Separation flotation machines are used. Four products are made, namely a lead-zinc concentrate, a clean zinc concentrate, a zinc middling, and an iron tailing. The lead-zinc concentrate is tabled on 32 Plat-O and Wilfley tables, a lead-iron concentrate being returned to the rake classifiers of the fine grinding plant; a clean zinc concentrate combined with that from the flotation cells, and a zinc middling to be re-treated. The zinc middling from the flotation cells and the zinc middling from the tables are reground in an 8-foot by 48-inch Hardinge ball mill. The iron tailing from the flotation cells is classified in a 10-foot bowl, 3-foot rake Dorr classifier, the sands being reground in the ball mill with the zinc middling, the slimes going to the dewatering plant. Zinc is recovered from the reground zinc middling and the sand tailing in an 8-cell 24-inch modified Minerals Separation flotation machine.

Dewatering plant.—The lead concentrate is dewatered in two modified Genter vacuum thickeners and in two 6-foot diameter, 6-disk, 2-valve American filters. The filter cake is conveyed to the lead concentrate loading bins. The zinc concentrate is dewatered in two 50 by 8-foot Dorr thickeners, and in three 6-foot diameter, 6-disk, 2-valve American filters. The filter cake is conveyed to the zinc loading bins. The iron tailing is dewatered in a modified Genter vacuum thickener and sluiced to the tailing pond.

Reagents used.—For the separation of the minerals, selective flotation is employed. The water used in the fine grinding plant is made alkaline by sodium carbonate. For the flotation of the lead sulphide, a mixture of water-gas tar and coal tar creosote is used with small amounts of sodium cyanide and cresylic acid. Copper sulphate and water-gas tar are added to the zinc cells for the flotation of the zinc sulphide. The consumption of reagents is considerably reduced by the return of solutions from the dewatering plant to storage tanks above the mill.

Special features of construction.—Ample room is provided on all floors. Floors are provided with overhead travelling cranes and are served by an inclined tramway running from the repair shops to the bottom of the mill. Individual direct connected motor drives are used on all machinery where practical. All circulating solutions and pulps are handled by Wilfley pumps. The arrangement for the pumps in the tunnel below the main floors in the centre of the building provides for taking care of spills, overflows caused by temporary shut downs, for keeping the mill clean, and delivering the material to the proper compartment of the mill. Railway track scales are provided for the weighing in of the ore as received from the coarse crushing plant, and the weighing out of the lead and zinc concentrates to the company's smelter.

Results being obtained.—In October, 1923, W. B. Timm, of the Department of Mines, visited the concentrator, and although it was not operating at capacity, due to shortage of power, 2,000 tons of ore were being treated daily. The full equipment was not being utilized; in the fine grinding plant, the bowl classifiers were not in operation; in the lead recovery plant, only one cleaner was being used; in the zinc recovery plant, the re-treatment portion for the zinc middlings, and the tables, were not in operation, yet a better grade of concentrate was being produced and a higher recovery obtained than in the concentrator at Trail, formerly treating this ore.

Working at full capacity, and with all units in operation, 3,000 tons of ore daily will be treated. Final concentrates made on November 17, 1923, showed the lead concentrate to contain—lead 70.1 per cent, zinc 5.4 per cent, and the zinc concentrate to contain—lead 4.0 per cent, zinc, 43.3 per cent.

THE SILVERSMITH CONCENTRATOR AT SANDON, B.C.

The Silversmith mill, of the Silversmith Mines, Ltd., Sandon, B.C., represents the latest design in modern mill practice for the concentration of the silver-lead-zinc ores of the Slocan district, by a combination of gravity methods and the oil flotation process.

The principal minerals of the Slocan ores are argentiferous galena, zinc blende, siderite (spathic iron) and iron pyrite. Tetrahedrite and chalcopyrite are found in small quantities associated, as a general rule, with the galena. A general assay of the Silversmith milling ore shows, silver 25 ounces per ton, lead 9 per cent, and zinc 7 per cent. The silver values being of greatest importance, the process that is followed recovers the greatest percentage of these values in a concentrate, having the greatest marketable value. The galena slimes badly, therefore the main object is to remove the greatest possible amount of it, by the use of jigs and tables, with as little grinding as possible. No attention is paid to the zinc blende in the gravity stages of concentration, as a clean zinc concentrate cannot be made by this means on account of the spathic iron having a similar specific gravity to the blende. A clean zinc concentrate is made by the flotation process after the galena is removed.

The concentrator has a daily capacity of 125 tons. The ore is supplied to the mill bins by an aerial tramway. It is crushed in a 10 by 20-inch Blake crusher, passed by a conveyer belt, which is also used as a sorting belt, to the mill feed bins. Clean ore is picked off the sorting belt and shipped direct to the smelter. From the mill bins the ore goes to a set of rolls, thence to jigs, and from there through successive stages of regrinding, classification, sizing, jig and table concentration, until finally reduced to a fineness (minus 80 mesh) suitable for treatment by flotation. Regrinding is done in two sets of rolls, a 4 by 5-foot cylindrical ball mill, and in a 6-foot by 16-inch Hardinge mill. Classification is done in two sets of 6-cell hydraulic classifiers, and in a Dorr classifier. Sizing is done in a set of 3 mm. trommels. By this method the galena is removed as soon as it is freed, without excessive loss in slimes from fine grinding. The jig and table concentrate is easily dewatered for shipment by a simple method of drainage.

The classifier overflows are thickened in Dorr settling tanks for flotation. The flotation plant is of the Callow pneumatic type, and consists of two lead rougher cells and one lead cleaner cell; one zinc rougher cell, and two zinc cleaner cells. Lead concentrate made from the two lead roughers goes to the lead cleaner, where a high-grade argentiferous lead concentrate is made, the tailing from the cleaner being returned to the head of the rougher cells. The tailing from the lead rougher cells goes to the zinc rougher cell, the tailing from which goes to waste and the concentrate passes on to the first cleaner cell. The concentrate from this cell is further cleaned in a second cleaner cell. The tailing from the cleaner cells is returned to the head of the zinc rougher cell. The final zinc

concentrate made is about 48 per cent zinc. The lead and zinc flotation concentrates are thickened in Dorr tanks and dewatered in American disk filters.

A visit was made to the mill by W. B. Timm, of the Department of Mines, in July, 1922. Since that time the concentrator has been in continual operation and regular shipments are made to the smelter of the Consolidated Mining and Smelting Co. of Canada, at Trail, B.C.

THE NEW BRITANNIA CONCENTRATOR AT BRITANNIA BEACH, B.C.

In January, 1923, the new concentrator for the concentration of the Britannia mine copper ores was completed. It was built with an initial capacity of 2,000 tons daily, but this tonnage can readily be increased to 3,000 tons by slight alterations in the fine grinding equipment. It replaces the former mill, destroyed by fire in 1921, and is of fireproof construction throughout. It is situated on a steep hillside, and has seven main floors for the various operations.

The Britannia ores are iron pyrite and chalcopyrite in a highly siliceous gangue. The metallic sulphides constitute about 17 per cent of the total weight of the ore.

Coarse crushing is done at the mine. The mine ore is reduced to 3-inch size in one 24 by 35-inch Blake crusher and in two Allis Chalmers No. 7½ gyratory crushers. From the coarse crushing plant, the ore is transported to bins of 2,500 tons capacity, situated above the concentrator at the beach.

The ore, as received from the mine, contains from 5 to 15 per cent moisture, mainly in the fines. A novel method of reducing the ore to three-sixteenths of an inch, for ball mill feed is practised without resorting to wet crushing, thus avoiding the difficulties generally experienced in dry crushing ore containing this percentage of moisture. The fines are washed out by spraying, the ore passing over Hummer screens having three-sixteenth-inch openings. Compressed air sprays are used on the coarse material which goes direct to the rolls containing only 2 per cent moisture. The fines, representing 20 per cent of the total ore, plus the wash water, go direct to the ball mills.

The roll plant consists of one 72-inch by 20-inch Traylor roll, crushing to three-quarters of an inch in closed circuit with 8 Hummer screens delivering oversize plus three-sixteenths of an inch to four 54-inch by 20-inch Traylor rolls in circuit with the screens. The ore reduced to three-sixteenths of an inch is conveyed to the fine storage bins, 3,600 tons capacity.

Fine grinding is done in eighteen 7 by 10-foot ball mills, each in closed circuit with a 19-foot simplex Dorr classifier. The feed from the fine storage bins is conveyed and divided by switch distributors to the classifiers. The ore is reduced to a fineness, 18 per cent on 65 mesh. The Dorr classifier overflows contain 20 per cent solids and are laundered direct to the flotation machines.

The flotation plant consists of six 14-cell Minerals Separation standard machines. The feed is distributed to the head of four of these units. Clean concentrate is obtained from the first four cells of each unit, a middling product is taken off from the remainder, and is conveyed by air lifts to the head of the other two units for recleaning, and the tailing is sent to

waste. From the two units used for recleaning the middling, a clean concentrate is obtained, and a middling which is returned to the head of the first four units.

The flotation concentrate is dewatered and settled in three 40-foot two tray Dorr tanks, the thickened concentrate being pumped to two 8-foot six disk American filters. The filter cake is conveyed a quarter of a mile to a 10,000-ton storage bin at the wharf. From this bin it is removed by cranes and conveyer belt to the steamers for shipment to the smelter.

A visit was made by W. B. Timm, of the Department of Mines, to the concentrator in October, 1923. At that time the mill feed was averaging 2.0 per cent copper, the copper concentrate produced contained 20 per cent copper, and the tailing contained 0.16 per cent copper. These results were being obtained by flotation of the chalcopyrite from the pulp, made alkaline by the addition of two pounds of lime per ton of ore. The flotation reagents used were a mixture of coal tar creosote and pine oil, and small quantities of sodium resinate. The original flow sheet of the concentrator included Hancock jigs ahead of the fine grinding mills for the removal of a portion of the copper as coarse concentrate. Danish flint pebbles were used in the fine grinding mills at the start. The flotation plant was installed for roughing only. Laboratory experiments showed that by changing from an acid to an alkaline pulp, the grade of the concentrate could be doubled by selective flotation of the chalcopyrite from the iron pyrite, without any appreciable loss of copper. The jigs were eliminated from the flow sheet; the pebble mills were changed over to ball mills for greater capacity, and the flotation units converted into rougher and cleaner units for selective flotation.

A NEW COPPER CONCENTRATOR AT ANYOX, B.C.

The Granby Consolidated Mining, Smelting, and Power Company is erecting a concentrator at Anyox, B.C., to concentrate the greenstone copper ores of the Hidden Creek ore-bodies. The initial capacity will be 1,500 tons daily. Crushing capacity of 3,000 tons is being installed to take care of increased capacity later.

The Hidden Creek ores are of two types, namely, a heavy pyritic ore, and a greenstone ore, impregnated with chalcopyrite and pyrrhotite. Smelting operations are semi-pyritic, so that in the course of mining operations a large tonnage of greenstone ore was left behind, it being too expensive to smelt this type of ore.

A typical analysis of the greenstone ore will show:—

Copper	1.85 per cent	Sulphur	16.0 per cent
Insoluble.....	42.0 "	Lime.....	4.0 "
Silica.....	35.0 "	Magnesia.....	5.0 "
Iron.....	24.0 "	Alumina.....	9.0 "

The chief sulphide minerals are chalcopyrite, pyrrhotite, pyrite, and sphalerite. Small amounts of precious metal values are also present in the ore.

The erection of the concentrator is the result of experimental work in the laboratory and in a pilot plant, in which 50,000 tons of ore were concentrated. It was proven that a copper concentrate, assaying from 12

to 15 per cent copper, with a concentration ratio of 8 to 1, and with a good recovery of the copper and precious metal values could be produced by selective flotation.

A visit was made to the works of the Granby Consolidated Mining, Smelting and Power Co., at Anyox, B.C., by W. B. Timm of the Department of Mines in November, 1923. At that time the foundations and floors of the concentrator were completed, the steel frame work was erected, and the machinery was being set in place. It was hoped to have the concentrator completed and ready for operation in the early spring. It is situated on the slope of a hillside between the mine and smelter. The mine ore will be reduced to 3-inch size in gyratory crushers and to three-eighth-inch size in two sets of Alaska type rolls. Fine grinding will be done in rod mills in closed circuit with drag classifiers, grinding to a fineness of 70 per cent through 200 mesh. The use of rod mills is new to fine grinding practice in this country, their use being confined to comparatively coarse grinding. The flotation plant will consist of Cole rougher and cleaner cells, the concentrates thickened in Dorr tanks and dewatered by Oliver filters.

THE COPPER CONCENTRATOR, ALLENBY, B.C.

The Allenby concentrator, built by the Canada Copper Corporation, was completed in 1919, and operated for a short time during the latter part of that year, when operations ceased, due to the decline in the copper market.

The 2,000-ton concentrator is situated five and a half miles south of the town of Princeton, B.C., and seven and a half miles farther south is the Copper Mountain mine, both connected by the Copper Mountain branch of the Kettle Valley railway from Princeton. Development work at the mine is reported to have blocked out 10,000,000 tons of copper ore of an average grade of 1.74 per cent copper.

The ore is chalcopyrite and bornite, disseminated through the granodiorite. Associated with it are small amounts of hematite, magnetite and iron pyrite. An average analysis of the ore is:

Copper.....	1.77	per cent	Sulphur.....	1.1	per cent
Silica.....	50.0	"	Alumina.....	19.9	"
Iron.....	5.8	"	Gold.....	0.005	oz./ton.
Lime.....	9.1	"	Silver.....	0.20	"

The ore lenses have been developed from three adits, the lowest one being used as the main haulage level. It is connected with the intermediate level by an ore pass 804 feet long. Below the lowest adit is the coarse crushing plant, consisting of one 30 by 42-inch jaw crusher in which reduction is made to 8-inch size, and two No. 7½ gyratory crushers reducing the ore to 2½ inch, from which it is delivered to storage bins of 2,000 tons capacity.

From the mine bins the ore is hauled in railway cars to concentrator bins of 1,800 tons capacity. From these bins it is conveyed and distributed to six 3 by 4-foot impact screens with five-eighth-inch openings, the oversize from the screens being crushed in one set of 72 by 20-inch Garfield rolls, three-quarter-inch opening, and conveyed to the five-eighth-inch impact screens. The undersize from the screens is conveyed to a 2,000-ton storage bin, from which it is drawn to two sets of 54 by 20-inch

rolls set close. After reduction by these rolls it is screened on eight Colorado Iron Works impact screens, 5-mesh openings, the oversize from the screens returned to the rolls, and the undersize going to the primary fine grinding units. Primary fine grinding is accomplished in four 7 by 10-foot ball mills, the discharge of the mills classified in four Dorr model D classifiers, the oversize ground in six 5 by 20-foot tube mills in closed circuit with six Akins classifiers. The overflow from the Dorr and Akins classifiers, which will practically all pass 80 mesh, a large proportion being finer than 200 mesh, goes to the flotation units.

From the flotation units, a concentrate, a middling that is returned to the head of the unit, and a tailing which goes to waste, are made. The concentrate is thickened in two 40-foot Dorr thickeners and dewatered in two continuous American filters, and the cake conveyed to a concentrate bin of 200 tons capacity.

The copper minerals, in the ore, being chalcopyrite and bornite, with only small amounts of iron sulphides present, permit of a high-grade copper concentrate, about 30 per cent copper being made by flotation. Such a concentrate is highly desirable for mixing purposes with lower grade ores and concentrates. The concentrate will be shipped to the Trail smelter of the Consolidated Mining and Smelting Co. of Canada.

A large pumping plant supplies water to the concentrator and other buildings from the Similkameen river. Electric power is furnished by the West Kootenay Power Co. from their transmission lines from Greenwood to Copper Mountain.

During 1923 the Granby Consolidated Mining, Smelting and Power Co. acquired the holdings of the Allenby Copper Co., until recently known as the Canada Copper Corporation, and took immediate steps towards putting the mine, surface plant and concentrator in condition for operation. Due to the unfavourable conditions of the copper market this work was suspended.

THE CONCENTRATOR FOR ROSSLAND ORES, TRAIL, B.C.

In November, 1923, the Consolidated Mining and Smelting Company of Canada, Ltd., commenced treating their Rossland ores in the concentrator at Trail, B.C., that had formerly been used for the treatment of the lead-zinc ores from the Sullivan mine. The new concentrator at Kimberley, B.C., having been put into successful operation on this ore, the Trail concentrator became available for the treatment of Rossland ores.

The original concentrator was built as a pilot plant for Rossland ores, and after running for some time, demonstrating the feasibility of treating these ores by flotation, it was re-adjusted and used as a pilot plant for the Sullivan ore. The successful treatment of the Sullivan ore having been demonstrated, it was enlarged to a daily capacity of 1,000 tons and treated Sullivan ore until the new concentrator at Kimberley was completed.

The Rossland ores consist of the sulphides of iron, pyrrhotite and pyrite, and chalcopyrite in a siliceous gangue. The average milling grade of the ore contains up to one per cent copper, with four to five dollars in gold values. Smelting operations on the crude ore being no longer profitable, a method of concentration prior to smelting, was developed after much experimental work.

The ore is transported from the mine bins at Rossland to Trail in standard railway ore cars. It is crushed in jaw crushers and rolls, and delivered to the concentrator bins. The concentrator has a capacity of 700 tons per day. Primary fine grinding is done in one 8-foot by 48-inch Hardinge ball mill, and in one 5 by 10-foot Marcy rod mill. Secondary fine grinding is accomplished in three 5 by 18-foot ball tube mills, each one being operated in closed circuit with a 6 by 25-foot Dorr rake classifier. A fineness 85 per cent through 200 mesh is obtained in the grinding circuit.

The pulp density for flotation is one of solids to three of water. A warm neutral circuit is maintained, the temperature being between 20° and 25°C. Two and one-half to three pounds per ton of coal tar creosote, and 0.05 per ton of pine oil or wood creosote are used.

Flotation is accomplished in two standard 24-inch Minerals Separation modified machines. Sub-aeration machines will later be used as cleaners. The final concentrate is thickened in two 40-foot Dorr thickeners and de-watered in two 4-foot 6-disk American continuous filters. The moisture in the cake is 14 to 15 per cent.

The feed averages 1 per cent copper and 0.25 oz. per ton gold. The concentrate made will average 8 per cent copper, 1.8 ozs. per ton gold. The tailing will average 0.06 per cent copper, 0.045 oz. per ton gold. The ratio of concentration is 1 : 8.5 and the recoveries are 93 per cent of the copper values, and 82.5 per cent of the gold values in the ore.

NEW CONCENTRATOR FOR THE EUSTIS MINING CO., EUSTIS, QUE.

Alterations to the concentrator of the Eustis Mining Company were completed and the concentrator put into operation in the early part of October, 1923.

The Eustis mine is in the Eastern Townships of Quebec, about 7 miles south of Sherbrooke. The first discovery of copper was made in 1865, and since then about a million tons of ore have been mined. The present shaft is now at a depth of over 4,000 feet.

The milling ore is a heavy sulphide, containing chalcopyrite associated with pyrite and small quantities of calcite, quartz, and other gangue minerals. It is free from arsenic and is an excellent ore for acid-making. The ore is now mined principally for the copper content. A typical analysis is copper 3 per cent, iron 40 per cent, sulphur 45 per cent, insoluble 7.5 per cent, other gangue 4.6 per cent.

Concentration by selective flotation.—Research carried on for a period of over a year, by the company in co-operation with the Department of Mines, was successful in working out a flotation process by which the copper-bearing mineral could be separated from the iron sulphides (pyrites) and gangue. The old concentrator, which had been able to do little more than eliminate the gangue minerals, was completely remodelled, and the new process installed.

A brief description of the new process is as follows. The ore is taken by electric haulage and gravity incline from the mine to the mill, where ample storage is provided. The mill has a capacity of 200 tons per day. Standard practice is used in the coarse crushing or breaking department of the mill, where the ore is reduced to one and one-half inches. A 300-

ton circular ore-bin provides storage capacity for the ball mill feed. The ore is fed by a belt conveyer to an 8-foot cylindrical ball mill, the discharge of this mill is divided between two 6-foot ball mills that operate in closed circuit with two duplex standard Dorr classifiers. The classifier overflows, which will pass a 100-mesh screen, are maintained at a high density (30 per cent solids) and elevated by Wilfley sand pumps to a Callow, flat bottom, flotation cell, which is 36 feet long by 24 inches wide and 18 inches deep. In this cell, the copper mineral is floated and the iron and gangue minerals dropped. A rougher concentrate, containing the copper, and an iron tailing, which goes to a separate section of the mill for the recovery of the iron sulphides (pyrites) is made. The rougher copper concentrate is recleaned in a second cell, from which a final copper concentrate and a middling product, which is returned to the first cell for re-treatment, are obtained. The final copper concentrate contains over 20 per cent copper and the recovery made is over 90 per cent.

Reagents for copper flotation.—The separation between the copper (chalcopyrite) and the iron sulphides is controlled by the use of an alkaline pulp. Lime at the rate of 5 to 10 pounds per ton is fed to the ball mills with the ore. The flotation oils used at present are neutral coal tar creosote and a mixture of 25 per cent thio-carbanilide and 75 per cent ortho-toluodine. A little, steam-distilled, pine oil is also used.

Pyrite recovery plant.—The pyrite in the ore is recovered for sale to the Nichols Chemical Co., at Capelton, one mile distant. The tailing from the copper flotation plant is dewatered in a large Dorr thickener to eliminate the bulk of the lime. The thickened pulp is mixed with fresh water and reduced to a pulp containing 25 per cent solids. This diluted pulp is acidified and distributed to two half-size Callow flotation cells, where the pyrite is recovered by flotation. The pyrite concentrate contains 50 to 55 per cent sulphur.

Reagents used for pyrite flotation.—The diluted pulp from the Dorr thickener is acidified by the addition of sulphuric acid. The quantity used is about 10 pounds per ton, original feed. The flotation oils used are fuel oil and crude pine oil.

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Canada Mines Bureau 57

3

CANADA
DEPARTMENT OF MINES
HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER
MINES BRANCH
JOHN MCLEISH, DIRECTOR

**INVESTIGATIONS IN ORE DRESSING
AND METALLURGY**

(Testing and Research Laboratories)

1924

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OTTAWA
F. A. ACLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
1926

No. 643

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HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER
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Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

Investigations of Mineral Resources and the Mining Industry.

Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).

Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).

Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.

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MINES BRANCH INVESTIGATIONS IN

ORE DRESSING AND METALLURGY, 1924

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the year 1924, the laboratories of the Division were devoted entirely to experimental and research work in connexion with the treatment of Canadian ores and metallurgical products. The results of the various investigations are given in the reports by the officials directly engaged on the work. Section I is a brief review of the investigations given in detail in Sections II, III, IV, and V, together with a list of the ores submitted for test purposes.

Section II (pages 8-91, inclusive)

The results of the investigations conducted in the Ore Dressing and Metallurgical Laboratory by Messrs. W. B. Timm, C. S. Parsons, R. K. Carnochan, and J. S. Godard, are included in this section. A brief review and summary of the investigations is as follows:—

Report No. 204 (pages 8-19, inclusive)

THE CONCENTRATION OF THE ZINC-LEAD ORES OF NOTRE DAME DES ANGES,
QUE., BY C. S. PARSONS

The experimental work was conducted on two small shipments of zinc-iron middlings, one small shipment of ore, and a carload shipment. The results obtained on the zinc-iron middlings indicated that these middlings could be concentrated by flotation, with a separation of the zinc from the iron, and the production of a marketable zinc product, with high recoveries of the zinc content. The preliminary results on the small ore shipment showed that the ore could be concentrated by selective flotation or by a combination of gravity concentration and selective flotation. The results obtained by tonnage check tests on the carload shipment verified the results obtained from the preliminary tests and showed that the ore could be readily concentrated, with the production of marketable lead and zinc products with high recoveries of the values in these metals, and furthermore that the silver and gold values, together with the small amount of copper mineral present in the ore, reported almost entirely with the lead product. A concentrating mill for the treatment of the ore has been built at the mine following a flow-sheet based on the results of these experimental tests.

Report No. 205 (pages 20-21, inclusive)

THE MANUFACTURE OF A WATERPROOFING COMPOUND FOR USE IN PAPER BOARD, BY W. B. TIMM

Waterproofing compounds for use in paper board have been manufactured from hot colloidal mixtures of various mineral substances. This report covers the preparation in the cold of a bentonite-asphalt mixture suitable for incorporating into felt, paper, or other materials.

Report No. 206 (pages 22-24, inclusive)

THE CONCENTRATION OF A LEAD-ZINC ORE FROM RIONDEL, B.C.,
BY R. K. CARNOCHAN

The ore was a mixture of lead and zinc sulphides with pyrite and arsenopyrite in a gangue of manganese carbonate, calcite, and quartz, carrying values in silver and gold. The results of the experimental tests indicated that the ore could be satisfactorily concentrated by selective flotation with a separation of the lead and zinc sulphides, producing marketable products of these minerals with good recoveries of the values.

Report No. 207 (pages 24-46, inclusive)

EXPERIMENTAL TESTS ON WINDPASS ORE, BY J. S. GODARD

The test work was conducted on two lots representative of the two types of ore from the Windpass mine, Chu Chua, B.C. The results indicate that the ore can be successfully treated by amalgamation, concentration by flotation, and cyanidation, with good recoveries of the values. The report covers in detail the various tests conducted on both types of ore, and on a mixture of the two types.

Report No. 208 (pages 46-49, inclusive)

THE CONCENTRATION OF GARNETIFEROUS GNEISS FROM BARRY'S BAY, ONT.,
BY R. K. CARNOCHAN

The experimental work conducted showed that wet concentration was preferable to dry concentration on this class of garnetiferous gneiss. It showed that a process of jigging and tabling would work very satisfactorily on the material, giving a product suitable to the requirements of the trade.

Report No. 209 (pages 49-51, inclusive)

GOLD ORE FROM MINE CENTRE, ONT., BY C. S. PARSONS

The test work conducted showed the ore very amenable to treatment by amalgamation followed by cyanidation, or by straight cyanidation. The ore submitted contained considerable free gold, and on this grade of ore amalgamation before cyanidation is recommended. If, however, the average grade of the milling ore was below one ounce to the ton, and very little coarse gold was present, amalgamation could be dispensed with and straight cyanidation used.

Report No. 210 (pages 51-53, inclusive)

THE CLEANING OF MICA PREPARATORY TO TRIMMING AND SPLITTING, BY
R. K. CARNOCHAN

The experimental tests show that the removal of the thin and small flakes by screening and suction is beneficial, giving a product more satisfactory for trimming and splitting, with a saving in hand labour. Much

of the fine particles adhering to the surfaces of the larger flakes is removed, permitting of a thinner portion being discarded in splitting. Washing tests were also made and the results obtained are given in the report.

Report No. 211 (pages 53-54, inclusive)

EXAMINATION OF MINERALIZED DIKE MATERIAL FROM LARDER LAKE, ONT.,
BY J. S. GODARD

Experimental tests were made to determine whether the gold values were in the arsenopyrite or in the pyrite. It was found that the values were associated with both minerals, but to a greater extent with the arsenopyrite.

Report No. 212 (pages 55-58, inclusive)

THE CONCENTRATION OF A GOLD-COPPER ORE FROM SPROAT LAKE, B.C.,
BY C. S. PARSONS

The experimental work showed the ore very amenable to concentration by flotation and table concentration. The gold values were found to be associated with the sulphides of copper and iron. A simple separation of the sulphides from the gangue was required. This could either be accomplished by classification, table concentration, and flotation, or by flotation and table concentration. The latter method was recommended.

Report No. 213 (pages 58-61, inclusive)

METALLURGICAL TESTS ON GOLD ORE FROM THE REX MINE, HERB LAKE,
MAN., BY C. S. PARSONS

The report on this ore covers the results obtained by amalgamation as against blanket concentration, for the recovery of the free gold. Table concentration and cyanide tests were also made. It was found that the ore was very amenable to treatment by amalgamation and cyaniding, and that amalgamation showed a higher recovery than blanket concentration for the recovery of the free gold.

Report No. 214-214a (pages 62-67, inclusive)

EXAMINATION AND EXPERIMENTAL TESTS ON GOLD ORE FROM DOMINION
CLAIMS, COPPER LAKE, MAN., BY W. B. TIMM AND J. S. GODARD

The ore was submitted to determine in what form the gold and silver values were present, and also to determine its amenability to treatment. It was found that the gold values were present both as free gold and tellurides; that the silver was present as ruby silver, and possibly as the sulphide, although this mineral was not detected by microscopic examination, and a small portion may be associated with the small amount of galena in the ore. The ore is amenable to treatment by cyanidation with high recoveries of the gold and silver values, and with low consumption of cyanide and lime.

Report No. 215 (pages 68-71, inclusive)

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE MOSS MINE, QUYON,
QUE., BY C. S. PARSONS

The report covers the concentration of two carload shipments of ore, and the results obtained prove conclusively the amenability of this type of molybdenite ore to concentration by flotation. A high-grade con-

centrate of 93 per cent MoS_2 was obtained, with recoveries of over 95 per cent of the molybdenite content in the ore. The correct conditions of operation to obtain these results are described in detail.

Report No. 216 (pages 71-74, inclusive)

EXAMINATION OF KIRKLAND LAKE MILL TAILINGS, BY C. S. PARSONS

The examination included screen analysis, classification tests, and analysis of the classified products, flotation tests, etc. The results of the screen tests and classification tests indicated that too coarse grinding was responsible to a large extent for the high tailing losses. No satisfactory results were obtained from flotation of the cyanide tailings.

Report No. 217 (pages 74-82, inclusive)

CONCENTRATION OF A SILVER-LEAD ORE FROM PORTLAND CANAL DISTRICT,
NEAR STEWART, B.C., BY C. S. PARSONS

The experimental work was of a preliminary nature indicating possible methods of treatment. These methods include selective flotation, with the production of both lead and zinc products, the silver reporting in both products, and straight flotation of the sulphides from the gangue. Cyanidation of the flotation tailings may be required.

Report No. 218 (pages 83-84, inclusive)

CONCENTRATION OF TITANITE (SPHENE) FROM PEGMATITE, BY C. S. PARSONS

A simple method of gravity concentration was followed for this material. The percentage of sphene in the rock was too low to make its separation of economic importance at the present time.

Report No. 219 (pages 85-87, inclusive)

EXPERIMENTAL TESTS ON CADWALLADER CREEK MILL TAILINGS, BY
J. S. GODARD

Examination of these tailings showed them to contain fairly high gold values. The test work included amalgamation, concentration, and cyanidation. Very good recoveries were made by cyanidation. As cyanidation was used for the recovery of the values in the Cadwallader Creek ores, it would seem that increased recoveries could be expected by more careful attention to the details of this practice.

Report No. 220 (pages 87-91, inclusive)

CONCENTRATION OF THE LEAD-ZINC ORE OF THE READER MINE, CALUMET
ISLAND, QUE., BY W. B. TIMM AND C. S. PARSONS

The experimental work showed that the ore was very amenable to treatment by selective flotation, with the production of high-grade lead and zinc products, with good recoveries. The tests also showed a good recovery of the silver values in the lead product.

Section III (pages 92-102, inclusive)

This section covers the progress of the work in the hydrometallurgical laboratory on the investigation being conducted by R. J. Traill and W. R. McClelland on the treatment of iron sulphide ore for the production of electrolytic iron, and the recovery of sulphur and other metals as by-products.

Section IV (pages 103-108, inclusive)

A brief summary of the work of the chemical laboratory is given in this section, together with a special investigation which was started during the year by H. C. Mabce, on the recovery of the iron and sulphur content, in addition to the nickel, copper, and precious metal values, from the nickeliferous pyrrhotite ores, and ores of a similar character.

Section V (pages 109-115, inclusive)

C. S. Parsons has prepared a short report which deals with the special problems connected with the concentration of lead-zinc ores of eastern Canada. Included in this section is a report, also by C. S. Parsons, on the concentration of the Lake George antimony ores, which is a summary of his experimental work in the laboratories.

List of Ores and Metallurgical Products on which Experimental Test and Research Work was Conducted

In the following table is given a list of the ores and metallurgical products received at the Ore Testing Laboratories for experimental test and research. The tabulated statement includes the report numbers, class of ore or product, source of shipment, name of shipper, and weight of shipment.

Report No.	Ore or product	Source of shipment	Shipper and address	Weight lbs.
204	Lead-zinc.	Notre Dame des Anges, Que.	Tetreault Bros., Notre Dame des Anges, Que.	54,242
205	Asphalt-bentonite.	Mansons, Ltd., Hawkesbury, Ont.	Mansons, Ltd., Hawkesbury, Ont.	500
206	Lead-zinc.	Riondel, B.C.	Shepherd Mining Co., Riondel, B.C.	177
207	Gold-copper.	Windpass mine, Chu Chua, B.C.	Windpass Mining Co., Chu Chua, B.C.	381
208	Garnet.	Barry's Bay, Ont.	Bancroft Mines Syndicate Ltd., Barry's Bay, Ont.	3,631
209	Gold.	Seine river, Mine Centre, Ont.	J. S. Hillyer, Esq., Mine Centre, Ont.	200
210	Mica.	Blackburn Bros., Ottawa, Ont.	Blackburn Bros., Ottawa, Ont.	564
211	Gold-arsenic.	Associated Goldfields, Larder Lake, Ont.	Dr. H. C. Cooke, Geological Survey, Ottawa	11
212	Gold-copper.	Sproat lake, near Alberni, B.C.	Messrs. Smith and Beavan, Alberni, B.C.	25
213	Gold.	Rex mine, Herb Lake, Man.	Manitoba Metals Co., Herb Lake, Man.	142
214	Gold-silver.	Dominion claims, Copper lake, Man.	A. L. Stewart, Esq., Winnipeg, Man.	205
215	Molybdenite.	Moss mine, Quyon, Que.	Canadian Wood Molybdenite Co., Quyon, Que.	135,200
216	Gold.	Kirkland Lake, Ont.	Wright Hargreaves Mines Ltd., Kirkland Lake, Ont.	50
217	Lead-zinc-silver.	Stewart, B.C.	S. G. O. Chalmers, Esq., Stewart, B.C.	54
218	Titanium.	Alleyn township, Pontiac district, Que.	Messrs. Spence and Cole, Mines Branch, Ottawa.	42
219	Gold.	Cadwallader creek, B.C.	A. W. Davis, Esq., Kamloops, B.C.	96
220	Lead-zinc.	Reader mine, Calumet island, Que.	W. B. Timm, Esq., Mines Branch, Ottawa.	50

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND
METALLURGICAL LABORATORY

Report No. 204

CONCENTRATION OF THE ZINC-LEAD ORES OF NOTRE DAME DES ANGES,
QUEBEC

C. S. Parsons

Shipments:—Four separate shipments were received from the property of Dr. J. L. A. Tetreault and associates of 730 Delorimier Ave., Montreal. The shipments were made up as follows:

Shipment No. 1 consisted of 100 pounds of zinc-iron middlings from a dump, produced from the operation of a gravity concentrator. This shipment was received in July, 1923.

Shipment No. 2 was a sample of the crude ore taken from the mill feed to the gravity concentrator then in operation. The shipment weighed 42 pounds and was received in July, 1923.

Shipment No. 3 consisted of 100 pounds of zinc-iron middling from tables in a gravity concentrator then in operation. This was received in July, 1923.

Shipment No. 4 was a carload of 27 tons of crude ore representing the mill feed to the gravity concentrator then in operation. This shipment was received in November, 1923.

Characteristics and Analyses of Shipments Nos. 1 and 3.—The zinc-iron middling product received consisted of a mixture of zinc and iron sulphides which had been freed by crushing to 40 mesh. Both pyrrhotite and pyrite were present, the pyrrhotite predominating. Analyses:

	Shipment No. 1	Shipment No. 3
Zinc.....	17.23 per cent	15.6 per cent
Lead.....	2.16 "	2.2 "
Iron.....	37.57 "	38.0 "
Gold.....	0.12 oz./ton	0.12 oz./ton
Silver.....	4.68 "	4.70 "

Characteristics and Analyses of Shipments Nos. 2 and 4.—These shipments represent the crude ore. The minerals are probably entirely freed by crushing to 65-mesh (Tyler standard screen). Mineralogically the ore is composed of galena, a dark zinc blende, pyrite, pyrrhotite, and a small amount of chalcopyrite in a siliceous gangue containing chloritic minerals and calcite. The galena is argentiferous and there is gold present, some of which is apparently in the free state. Analyses:

	Shipment No. 2	Shipment No. 4
Zinc.....	6.43 per cent	5.59 per cent
Lead.....	1.86 "	2.53 "
Iron.....		6.78 "
Gold.....	0.08 oz./ton	0.06 oz./ton
Silver.....	2.98 "	3.75 "

Object of Experimental Work.—The property from which these samples were taken has been operated from time to time for a number of years. Numerous attempts have been made to separate and recover both a lead and zinc concentrate which could be marketed. The attempts to produce a zinc concentrate were failures, but the mine was operated, and is being operated at present, to produce a lead concentrate. The lead is concentrated by graded crushing and tabling. Both flotation and magnetic separation have been tried on the ore, and mills were built to use these processes. During the war period a magnetic plant operated for some time, but it is reported that it was not a success. When flotation was used, the iron and zinc were both floated, the bulk of the lead having been previously eliminated by tables. The flotation concentrate of zinc and iron was given a magnetic roast and the iron eliminated. This process was not a commercial success.

The purpose of the experimental work in the case of the shipment of zinc-iron table middlings taken from the dump produced by previous milling operations, was to determine whether a successful method could be worked out whereby a high-grade zinc concentrate could be produced. Shipment No. 3 was the same material as No. 1 except that it was freshly produced middling, whereas the material of shipment No. 1 had lain exposed to the weather for some time. The purpose of submitting shipment No. 3 was to determine whether there was any material difference between the two products with respect to their response to various methods of separation.

Shipments Nos. 2 and 4 which consisted of the crude ore were submitted to determine whether a possible method could be worked out so that both a lead and zinc concentrate could be recovered as marketable products.

EXPERIMENTAL TEST ON SHIPMENTS NOS. 1 AND 3

The work done on the shipments of zinc-iron middlings was confined to small-scale magnetic and flotation tests.

Magnetic Separation Tests.—Two methods were tried: separation of the raw middlings, and of middlings which had been given a flash roast to change the pyrite to the magnetic sulphide.

Results.—No results were obtained by the first method. By the second method the results were very promising. The following is typical of a number of tests made:—

Product	Weight per cent	Analysis					Per cent of zinc values
		Zn %	Pb %	Fe %	Au. oz./ton	Ag. oz./ton	
Non-magnetic zinc.....	41.0	39.91	4.36	8.3	0.04	4.76	91.2
Magnetic iron.....	59.0	2.69	0.87	54.5	0.05	4.10	8.8

Conclusions.—Several adjustments were made to the magnetic separator used, but very little difference in either grade or recovery was obtained. The zinc product averaged about 39 per cent with a recovery of 93 per cent. An exceptionally good elimination of the iron was obtained. The

reason why a higher grade zinc product was not obtained was because the gangue minerals in the feed reported in the non-magnetic product. These could probably be eliminated by air jigs or tables, and the grade of the zinc product increased.

Flotation Tests.—Only small-scale batch flotation tests were made on the middlings. The flotation of the middling product was given careful consideration, particularly the dump material. To float a zinc sulphide from such a product involves a difficult problem in selective flotation. A zinc concentrate was required which would contain 42 to 45 per cent zinc, the tailing consisting almost entirely of iron sulphide.

General Procedure.—The middlings were ground in a small ball mill to 65 mesh. The density of the pulp in the mill was 1 : 1. A charge was prepared for each test. Some of the reagents were added to the ball mill and others directly to the flotation cell.

Results of Flotation Tests.—Very promising results were obtained. The following is typical of a number of tests made:—

Shipment No. 1—Old Dump Middling

Product	Weight per cent	Analysis					Per cent of zinc values
		Zn %	Pb %	Fe %	Au. oz./ton	Ag. oz./ton	
Concentrate.....	36.6	42.08	4.69	13.23	0.16	9.90	90.4
Middling.....	7.5	11.60	1.75	45.85	0.02	4.20	5.1
Tailing.....	55.9	1.25	trace	52.92	0.03	1.65	4.5

Shipment No. 3—Freshly Produced Middling

Product	Weight per cent	Analysis					Per cent of zinc values
		Zn %	Pb %	Fe %	Au oz./ton	Ag. oz./ton	
Concentrate.....	34.1	42.15	0.10	12.74	92.1
Middling.....	6.0	7.99	0.12	11.52	3.1
Tailing.....	59.9	1.25	0.02	4.48	4.8

Conclusions.—Although the results of these tests could very probably be duplicated in actual practice on the freshly produced middling, difficulty might be encountered in the case of the old dump material. Small batch tests have been found to give in some cases unreliable results on such material. It would be necessary to run large-scale tonnage check tests on a truly representative shipment of the dump material before a definite opinion could be given on this method of treatment.

EXPERIMENTAL TESTS ON SHIPMENT No. 2

Flotation Test.—Only one test was made on the ore of this shipment which represents the crude ore. The results are given in the following table:—

Product	Weight per cent	Analysis			Per cent of zinc values
		Zn %	Au. oz./ton	Ag. oz./ton	
Concentrate.....	14.8	41.3	0.12	16.46	81.6
Middling.....	9.2	12.5	0.12	10.60	15.4
Tailing.....	76.0	0.3	0.04	0.66	3.0

Conclusions.—A high-grade zinc concentrate can be obtained from the crude ore by flotation and a good recovery made without any apparent difficulty.

SMALL-SCALE EXPERIMENTAL TESTS ON SHIPMENT No. 4

This shipment consisted of a carload of crude ore from the mine. Small-scale selective flotation tests were run to determine whether it was possible to make a separation between the lead and zinc minerals and to produce a marketable concentrate of each.

General Procedure.—The ore was ground in a ball mill to 65 mesh. The density of the pulp in the mill was 1 : 1. For the flotation of the lead some of the reagents were added to the ball mill; others directly to the flotation cell. The lead tailing was dewatered, the pulp density raised to 1 : 3 by the addition of fresh water, and the reagents for the flotation of the zinc added to the flotation cell.

Results of Selective Flotation Tests.—The results obtained are given in the following table:—

Product	Weight per cent	Analysis				Per cent of values	
		Zn %	Pb %	Au oz./ton	Ag oz./ton	Zn	Pb
Lead concentrate.....	8.6	7.52	24.42	0.32	32.48	12.9	93.5
Zinc concentrate.....	9.7	40.45	0.51	trace	1.38	78.4	2.2
Middling.....	5.5	4.43	0.41	0.02	1.44	4.9	1.0
Lead concentrate.....	4.8	4.12	25.44	0.28	34.86	4.1	55.5
Zinc concentrate.....	10.5	40.87	7.50	0.04	9.08	86.0	35.8
Zinc middling.....	6.5	4.92	1.18	0.22	5.12	6.4	3.5
Tailing.....	78.2	0.21	0.15	trace	0.24	3.4	5.3
Lead concentrate.....	7.7	8.49	34.47	10.9	90.5
Zinc concentrate.....	15.1	34.94	1.13	83.5	5.8
Tailing.....	77.2	0.47	0.14	5.7	3.7

Conclusions.—The results of these tests were very satisfactory, showing the possibility of selective flotation of the ore with the production of marketable lead and zinc concentrates. In conducting the tests, the lead concentrate was not recleaned, and in the last test given above, the zinc concentrate was not recleaned.

TONNAGE CHECK TESTS ON SHIPMENT No. 4

Nine tonnage check tests were run on the ore of shipment No. 4 in order to obtain more reliable data on the treatment of the ore. The procedure and results of the tests are summarized and given below:

Test No. 1

General Description of Test.—The ore was crushed to $\frac{1}{2}$ inch in a jaw crusher and set of rolls. The $\frac{1}{2}$ -inch material was fed to a $4\frac{1}{2}$ -foot Hardinge ball mill, first passing through a Vezin sampler where $\frac{1}{20}$ was cut out for a head sample. The ball mill was operated in closed circuit with a 15-inch standard Dorr classifier. The classifier overflow which would pass 48 mesh went direct to two 6-foot by 12-inch Callow flat-bottom flotation cells where a rougher concentrate and final tailing were made. The rougher concentrate was re-cleaned twice in two 3-foot by 12-inch cleaners. The cleaner tailing was returned to the head of the roughers by an air lift. The heavy tar oil was fed to the ball mill, but the soluble oils and pine oil were fed at the classifier overflow. The soda ash and lime were added to the ball mill. The final concentrate was passed over a Wilfley table, using a standard deck.

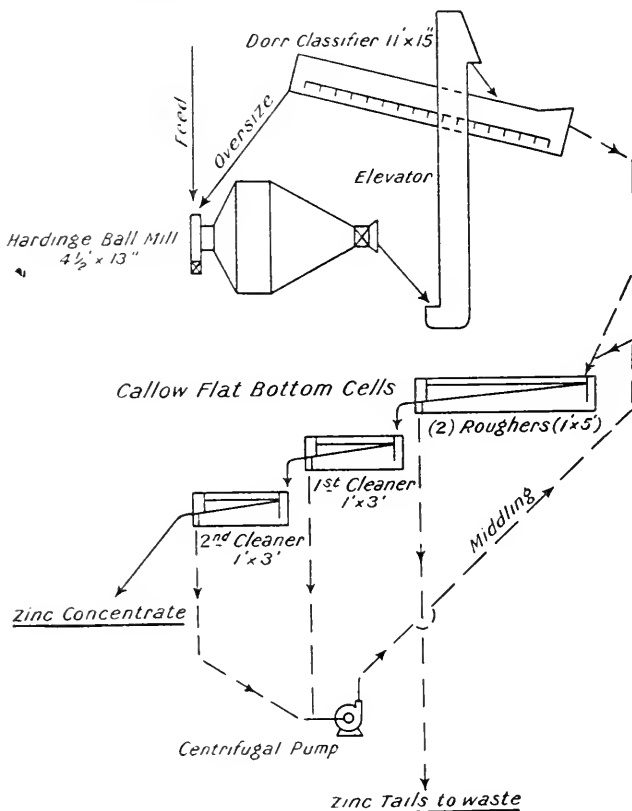


Figure 1. Flow-sheet, Shipment No. 4, Test No. 1.

Rate of feed per hour, 1,050 pounds.

Oils used were Barrett Co.'s. water-gas tar, General Engineering Co.'s TT mixture, and steam-distilled pine oil from Hercules Powder Co.

Analyses of Samples

Product	Zinc %	Lead %	Iron %	Gold oz./ton	Silver oz./ton
Head sample (Vezin sampler).....	5.28	2.13	6.93	0.08	3.90
Classifier overflow (feed to cells).....	5.78	1.86	4.68	0.03	2.60
Concentrate sample No. 1.....	30.88	7.61	19.31	0.08	13.68
Tailing, sample No. 1.....	0.78	0.28	3.05	0.02	0.38
Concentrate, sample No. 2.....	32.29	11.32	15.26	0.03	13.17
Tailing, sample No. 2.....	0.58	0.28	4.06	0.02	0.80
Table concentrate (lead).....	2.25	68.78	0.36	62.04
Table tailing (zinc).....	32.48	5.88	0.03	11.31

Recapitulation of Results Using Classifier Overflow as Head Sample

Zinc—	Average grade of concentrate.....	32.48	per cent
	Average grade of tailing.....	0.68	"
	Total recovery of zinc in feed.....	90.30	"
Lead—	Grade of table concentrate.....	68.78	"
	Total recovery of lead in feed.....	44.9	"
	Recovery of lead by tabling flotation concentrate.....	51.0	"

Summary of Distribution of Lead in Products

Lead lost in zinc product.....	43.1	per cent
Lead lost in flotation tailing.....	12.0	"
Lead recovered in table concentrate.....	44.9	"

Test No. 2

General Description of Test.—The ore was crushed to $\frac{1}{2}$ -inch and a head sample cut out by a Vezin sampler. The ore after passing through the sampler was fed to a 4 $\frac{1}{2}$ -foot Hardinge ball mill in closed circuit with a Dorr classifier. The classifier overflow which would pass 48 mesh was passed over a Wilfley table where a lead concentrate was made. The table tailing carrying the zinc was dewatered in an 8-foot Callow cone, and the thickened feed went to two 6-foot by 12-inch Callow flat-bottom cells, each cell receiving half the feed. As in the previous test the final tailing was produced from these two rougher cells and the rougher concentrate was recleaned twice in two 3-foot by 12-inch cleaner cells, the cleaner tailing being returned to the rougher cells. The final flotation zinc concentrate was tabled on a Wilfley table to remove a second lead concentrate.

Rate of feed per hour, 1,200 pounds.

Oils used were Barrett's No. 634, TT mixture and YZ mixture.

Steam-distilled pine oil.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Gold oz. per ton	Silver oz. per ton
Whole.....	Head sample (Vezin sampler).....	5.78	2.68	0.05	3.97
	Classifier overflow (table feed).....	5.88	2.15	0.02	3.78
	Table concentrate, lead.....	1.93	71.41	6.5	0.70	75.60
10.30- 1.00	Thickened feed to cells, sample No. 1.	5.48	0.72	0.01	1.99
10.30-12.00	Flotation concentrate, sample No. 1....	25.68	2.58	trace	6.40
10.30-12.00	Flotation tailing, sample No. 1.....	0.46	0.21	trace	0.54
12.00- 1.05	Flotation concentrate, sample No. 2....	45.55	4.78	0.10	10.90
12.00- 1.05	Flotation tailing, sample No. 2.....	0.81	0.21	trace	0.60
1.00- 3.00	Thickened feed to cells, sample No. 2.	6.08	1.23	0.01	2.59
1.05- 1.30	Flotation concentrate, sample No. 3....	38.95	4.97	0.10	11.60
1.05- 1.30	Flotation tailing, sample No. 3.....	0.25	0.31	trace	0.40
1.30- 3.00	Flotation concentrate, sample No. 4....	36.18	4.92	12.4	0.10	10.70
1.30- 3.00	Flotation tailing, sample No. 4.....	0.35	0.31	trace	0.35
Whole.....	Flotation concentrate.....	35.80	6.20	0.10	11.60
"	Table concentrate, lead, No. 2.....	18.19	25.86	0.40	41.60
"	Table tailing, final zinc concentrate....	37.89	3.49	0.04	6.56
	Average of lead, table concentrate.....	7.78	55.0

Recapitulation of Results using Classifier Overflow as Head Sample

Average grade of lead concentrate.....	55.0	per cent
Total recovery of lead values.....	72.73	"
Recovery of lead values from flotation concentrate.....	50.4	"
Average grade of zinc concentrate.....	37.89	"
Average recovery of zinc values.....	93.75	"

Summary of Distribution of Lead Values in Products

Lead recovered by first tabling.....	55.6	per cent
Lead recovered by tabling flotation concentrate.....	17.13	"
Lead values lost in flotation tailing.....	10.3	"
Lead values lost in zinc concentrate.....	16.97	"

Test No. 3

General Description of Test.—The same procedure was followed as in test No. 2.

Rate of feed per hour, 1,500 pounds.

Reagents Used.—A fuel oil manufactured by the G.N.W. Oil Co., Cleveland, Ohio, was tried, but found to be of no use, as both a dirty concentrate carrying iron and a high tailing were made. During the operations it was noticed that considerable gangue was floating with the zinc. An analysis was made on concentrate sample No. 4 to determine whether it was the iron or the gangue that was reducing the grade of the concentrate.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Gold oz.	Silver ton
Whole.....	Head sample (Vezin sampler).....	5.86	2.57	0.05	3.61
"	Classifier overflow.....	5.56	2.46	0.02	3.68
"	Table concentrate (lead).....	3.29	63.62	0.60	66.9
"	Table tailing.....	6.28	1.23	trace	1.78
10.35-11.40	Flotation concentrate (zinc), sample No. 1.....	40.58	7.50	0.10	12.9
10.35-11.40	Flotation tailing, sample No. 1.....	1.30	0.41	trace	0.74
11.40-12.40	Flotation concentrate (zinc), sample No. 2.....	42.78	7.39	0.10	14.20
11.40-12.40	Flotation tailing, sample No. 2.....	0.82	0.36	trace	0.62

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Gold oz. per ton	Silver oz. per ton
12.40- 1.40	Flotation concentrate (zinc), sample No. 3.....	27.95	7.71	0.10	13.8
12.40- 1.40	Flotation tailing, sample No. 3.....	2.63	0.31	trace	0.76
1.40- 2.10	Flotation concentrate (zinc), sample No. 4.....	41.97	5.85	0.10	11.9
1.40- 2.10	Flotation tailing, sample No. 4.....	2.11	0.51	trace	0.92
2.30-3.45	Flotation concentrate (zinc), sample No. 5.....	42.80	7.70	0.10	15.8
2.30-3.45	Flotation tailing, sample No. 5.....	2.26	0.62	trace	1.42
Whole.....	Combined flotation concentrate.....	38.75	8.72	9.6	0.10	15.48
	Table concentrate No. 2 lead.....	8.02	62.79	0.55	68.55
	Table tailing, final zinc concentrate....	41.36	4.92	0.08	10.46

Recapitulation of Results using Classifier Overflow as Head Sample

Average grade of lead concentrate.....	63.4 per cent
Total recovery of lead values.....	66.7 "

The recoveries of the zinc values varied according to the changes made in the use and regulation of the flotation reagents. Recovery based on sample No. 2 obtained with 0.20 pound per ton No. 634 oil and 0.15 pound YZ mixture:—

Zinc concentrate.....	42.78 per cent
Recovery of zinc.....	88.6 "

Summary of Distribution of Lead Values in Products

Recovery of lead in first table concentrate.....	51.0 per cent
Recovery of lead in second table concentrate.....	15.7 "
Loss of lead in zinc concentrate.....	17.5 "
Loss of lead in tailing.....	15.8 "

Test No. 4

General Description of Test.—The ore was crushed to 1 inch and a head sample cut by a Vezin sampler. The ore after passing through the sampler was fed to a 4½-foot Hardinge ball mill in closed circuit with a Dorr classifier. The classifier overflow which would pass 48 mesh was passed over a Wilfley table where a lead concentrate was made. The table tailing was thickened in a Callow cone and the thickened feed went to two 6-foot by 12-inch Callow flat-bottom cells. A final tailing and a rougher concentrate were made. The rougher concentrate was recleaned in two 3-foot by 12-inch cleaner cells, the cleaner tailing being returned with the feed to the rougher cells. This zinc flotation concentrate was tabled to remove as much lead as possible. Rate of feed per hour, 1,000 pounds. Reagents used, same as in Test No. 2.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Insol. %	Gold oz. per ton	Silver oz. per ton
Whole.....	Head sample (Vezin sampler).....	5.98	2.36	6.46	0.07	3.53
"	Head sample (Classifier overflow).....	5.54	2.77	6.77	0.05	3.41
"	Table concentrate No. 1, lead.....	2.27	58.5	13.74	0.90	68.3
"	Table tailing after thickening.....	5.68	1.85	7.37	0.03	2.55
11.50-1.20	Flotation concentrate, sample No. 1.....	44.03	7.28	9.09	1.51	0.10	10.8
11.50-1.20	Flotation tailing, sample No. 1.....	1.19	0.36
1.20-3.30	Flotation concentrate, sample No. 2.....	43.9	7.39	8.89	2.76	0.10	13.98
1.20-3.30	Flotation tailing, sample No. 2.....	1.76	0.41
3.30-4.00	Flotation concentrate, sample No. 3.....	42.25	10.18	8.80	4.90	0.10	12.46
3.30-4.00	Flotation tailing, sample No. 3.....	0.95	0.40	6.4	trace	0.58
4.00-4.45	Flotation concentrate, sample No. 4.....	43.0	17.25	8.30	4.7	0.12	14.12
4.00-4.45	Flotation tailing, sample No. 4.....	1.08	0.30	6.4	trace	0.08
Whole.....	Flotation concentrate.....	44.0	9.23
"	Table concentrate No. 2, lead.....	9.26	63.8	4.85	0.90	76.6
"	Table tailing, final zinc concentrate.....	44.13	7.28	9.4	1.51	0.12	12.36

Recapitulation of Results using Classifier Overflow as Head Sample

Lead—Average grade of concentrate..... 59.1 per cent

Zinc—The recovery varied during the test:—

Sample No. 1	Recovery of zinc.....	80.7	per cent
" " 2	"	71.4	" "
" " 3	"	85.2	" "
" " 4	"	82.6	" "

The lead in the flotation concentrate from this test was so fine that a poor recovery was made on the table.

Test No. 5

General Description of Test.—The procedure followed was similar to that of the preceding tests. The zinc concentrate was not tabled for the removal of the lead as sufficient data had already been collected on this operation. In figuring the recovery of the lead, a recovery of 50 per cent of the lead in the flotation concentrate was assumed as this figure corresponds with the results obtained in the preceding tests. Rate of feed per hour, 1,200 pounds. Oils used same as in former tests.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %
Whole.....	Head sample (Vezin sampler).....	5.68	3.18	6.72
"	Head sample (classifier overflow).....	5.17	2.05	6.00
"	Table concentrate No. 1, lead.....	2.69	59.84	11.00
"	Table tailing (feed to cells).....	4.81	0.92	5.30
11.30-12.45	Flotation concentrate, sample No. 1.....	43.53	6.67	9.39
11.30-12.45	Flotation tailing, sample No. 1.....	0.42
	Mill shut down 12.45-3.20.
3.20- 3.45	Flotation concentrate, sample No. 2.....	42.39	6.06	10.86
3.20- 3.45	Flotation tailing, sample No. 2.....	0.23
3.45- 4.10	Flotation concentrate, sample No. 3.....	45.51	5.54	9.29
3.45- 4.10	Flotation tailing, sample No. 3.....	0.36
4.10- 4.40	Flotation concentrate, sample No. 4.....	42.81	6.98	9.24
4.10- 4.40	Flotation tailing, sample No. 4.....	0.47
Whole.....	Average flotation tailing (calculated).....	0.37
"	Final flotation concentrate.....	42.39	6.78	10.00

Recapitulation of Results using Classifier Overflow as Head Sample

Lead: Average grade of concentrate.....	59.84 per cent
Average recovery assuming 50 per cent lead in flotation concentrate recovered by tabling.....	78.0 "
Zinc: Average grade of zinc concentrate (not tabled to remove lead).....	42.4 "
Average recovery for whole test.....	93.6 "

Test No. 6

General Description of Test.—This test was a selective flotation test. The ore was crushed as in the previous tests and the classifier overflow which would pass 35 mesh was fed to a four-cell Ruth flotation machine where the lead was floated. The tailing from the Ruth went direct to the Callow cells where the zinc was floated. Rate of feed per hour, 1,200 pounds.

Reagents for flotation of lead: sodium cyanide, oil mixture, cresylic acid.

Reagents for flotation of zinc: water-gas tar, copper sulphate, No. 634 oil, and pine oil No. 5.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Insoluble %
Whole.....	Head sample (Vezin sampler).....	5.65	2.87	6.87
"	Classifier overflow, head sample.....	6.10	2.36	6.46
"	Lead concentrate (Ruth machine).....	4.65	35.51	6.72	28.3
"	Lead tailing—feed to zinc flotation.....	5.26	0.21	6.00
1.15-2.15	Zinc flotation concentrate, sample No. 1.....	50.66	0.61	10.91
1.15-2.15	Zinc flotation tailing, sample No. 1.....	1.96	0.10
2.15-3.00	Zinc flotation concentrate, sample No. 2.....	51.18	0.72
2.15-3.00	Zinc flotation tailing, sample No. 2.....	7.64	0.11
3.00-3.45	Zinc flotation concentrate, sample No. 3.....	50.05	0.92
3.00-3.45	Zinc flotation tailing, sample No. 3.....	2.53	0.13
3.45-4.20	Zinc flotation concentrate, sample No. 4.....	48.02	0.92
3.45-4.20	Zinc flotation tailing, sample No. 4.....	1.24	0.11
Whole.....	Final zinc concentrate.....	50.04	0.72
"	Final zinc tailing, average.....	1.76

Recapitulation of Results using Classifier Overflow as Head Sample

Lead: Average grade of concentrate.....	35.51 per cent
Average recovery.....	90.0 "
Zinc: Average grade of concentrate.....	50.09 "
Average recovery.....	73.8 "

Note.—At times during test, the recovery was over 85 per cent.

Test No. 8

General Description of Test.—This test, together with test No. 9, was run to demonstrate the two methods of concentration for Messrs. Tetreault and their engineer, who were present by appointment. The procedure followed in this test is similar to that used in tests Nos. 2 to 5. Rate of feed per hour, 1,000 pounds.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %
Whole.....	Head sample (Vezin sampler).....	4.92	1.96	7.02
"	Classifier overflow (feed to table).....	4.87	1.62
"	Table concentrate (lead).....	10.85	68.26
"	Table tailing (feed to cells).....	4.42	0.45
12.20-2.30	Flotation concentrate, sample No. 1.....	38.65	5.12
12.20-2.30	Flotation tailing, sample No. 1.....	0.48	0.22
Whole.....	Flotation concentrate.....	37.65	4.76

Recapitulation of Results using Classifier Overflow as Head Sample

(Assuming that by re-tabling the zinc concentrate 50 per cent of the lead in it would be recovered)

Lead: Average grade of concentrate.....	68.26 per cent
Average recovery of lead.....	78.80 "
Zinc: Average grade of concentrate.....	37.65 "
Average recovery of zinc.....	91.4 "

Test No. 9

General Description of Test.—This was a selective flotation test. The object was to produce a high-grade lead and zinc concentrate. The classifier overflow which would pass 48 mesh was fed directly to a four-cell Ruth flotation machine where a lead concentrate was made in one operation, no cleaner cell being used. The concentrates from the last two cells were returned to the feed end of the machine, a finished concentrate being taken from the first two cells. The tailing was pumped to Callow cells consisting of two rougher cells operated in parallel, and two cleaners in parallel. The middling product from the two cleaners was returned to the feed end of the rougher cells. Rate of feed per hour, 1,400 pounds.

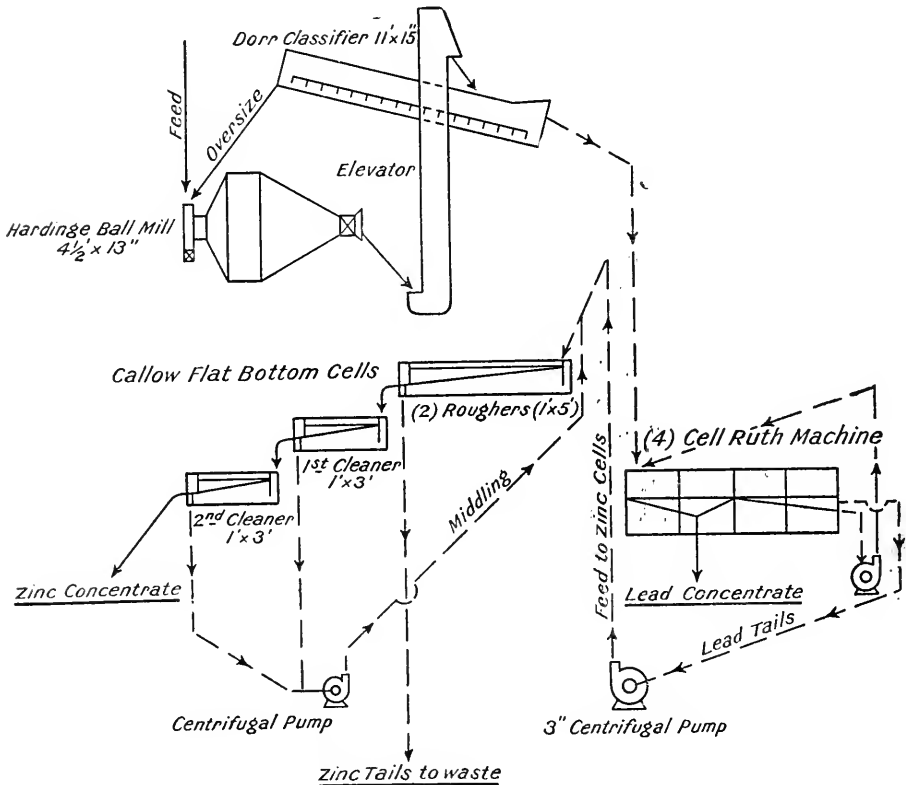


Figure 2. Flow-sheet, shipment No. 4, Test No. 9.

Lead flotation reagents: soda ash, sodium cyanide, cresylic acid and oil mixture.

Zinc flotation reagents: copper sulphate, No. 634 oil, TT mixture, and pine oil No. 5.

Analyses of Products

Product	Zinc %	Lead %	Iron %	Insol. %	Gold oz/ton	Silver oz/ton
Classifier overflow (head sample).....	5.52	1.96	7.42	0.02	3.28
Lead concentrate, sample No. 1.....	4.17	51.09	2.60	16.13	0.30
Lead tailing, sample No. 1.....	5.12	0.22	6.16	49.04	tr.	0.7
Zinc concentrate, sample No. 1.....	41.46	2.72
Zinc tailing, sample No. 1.....	0.15	0.22	9.39
Zinc concentrate, sample No. 2.....	47.18	1.15
Zinc tailing, sample No. 2.....	0.53	0.17
Zinc concentrate, total sample.....	46.19	1.09
Zinc tailing, total sample.....	0.30	0.15
Special sample first cleaner, zinc concentrate.	20.86	1.29	11.92	11.30
Special sample of table concentrate by tabling final flotation tailing.....	3.64	0.06	6.44

Recapitulation of Results using Classifier Overflow as Head Sample

Average grade of lead concentrate.....	51.09 per cent
Average recovery of lead.....	89.4 "
Average grade of zinc concentrate.....	46.17 "
Average recovery of zinc.....	94.8 "

GENERAL SUMMARY AND CONCLUSIONS

Two methods of concentration are suggested for the treatment of this ore, namely tabling to recover the lead followed by flotation of the table tailing to recover the zinc and re-tabling the flotation zinc concentrate to recover additional lead. The alternative method is selective flotation, whereby a slightly lower grade lead concentrate is obtained, but with a much higher recovery than by the first method, and a higher grade zinc concentrate, lower in lead, with approximately the same recovery as by the first method.

By the first method, provided that a system of graded crushing, screening, and classification is used, and a classified product fed to the tables, and the flotation concentrate is re-tabled, 80 per cent of the lead values should be recovered in a high-grade lead concentrate containing 65 to 70 per cent lead. The gold and silver values in the ore are practically all recovered and chiefly report in the lead concentrate. The zinc concentrate consistently contains only 0.10 ounce of gold, and approximately 12.0 ounces of silver per ton. The zinc recovery is high, and 90 per cent can be expected with a grade of concentrate assaying from 43 to 45 per cent zinc. The chief difficulty in obtaining a high-grade zinc concentrate is due to the tendency of the chloritic gangue mineral to float. This may not be so marked in a larger flotation unit.

By the second method of selective flotation, a 50 per cent lead concentrate can be obtained with a 90 per cent recovery of the lead values, and containing, from figures obtained in small-scale flotation test No. 1, 95 per cent of the gold and 87 per cent of the silver. A 50 per cent zinc concentrate can be obtained, having a lead content of less than 1.5 per cent and with a recovery of better than 90 per cent of the zinc values.

Report No. 205

EXPERIMENTAL TESTS ON THE USE OF BENTONITE IN THE MANUFACTURE OF WATERPROOFING COMPOUND FOR PAPER BOARD

W. B. Timm

Shipment.—Several small lots of asphalt, china clay, bentonite, and resin, were received March 5, 1924 from Mansons Limited, Hawkesbury, Ont.

Purpose of Experimental Tests.—The experimental tests on the above material were conducted to determine whether an asphaltic waterproofing compound could be obtained by grinding a mixture of these substances in the cold, suitable for incorporating into felt, paper, or other materials.

Preliminary Preparation of Material.—The asphalt was first crushed to $\frac{1}{2}$ inch in a small jaw crusher and the bentonite ground to 100 mesh in a Raymond pulverizer. Bentonite from two sources was used separately, for comparison, one lot being from Rosedale, Alberta, and the other from Medicine Bow, Wyoming; the latter was supplied by the Owyhee Chemical Products Co., Chicago, being the product marketed as "Wilkinite." No material difference was perceptible in the products made from the two bentonites. Extremely fine preliminary grinding of the bentonite is not necessary, nor would it be necessary to crush the asphalt to $\frac{1}{2}$ inch for large-scale grinding operations.

EXPERIMENTAL TESTS

The test work consisted of dry and wet grinding of the asphalt-bentonite, asphalt-china clay, and resin-bentonite mixtures, using various grinding devices.

Test No. 1

Mixture: 80 per cent petroleum asphalt
20 per cent bentonite

Ground in a 12 by 12-inch ball mill for $2\frac{1}{2}$ hours with twice the weight of water. Charge consisted of 4 pounds asphalt, 1 pound bentonite, and 10 pounds water.

Result: A fine homogeneous, gelatinous mass which when washed on a 200-mesh screen gave 20 per cent coarser than 200 mesh.

Test No. 2

Mixture: 85 per cent petroleum asphalt
15 per cent bentonite

Ground in a 12 by 12-inch ball mill for $2\frac{1}{2}$ hours with $1\frac{1}{2}$ times the weight of water. Charge consisted of $4\frac{1}{4}$ pounds asphalt, $\frac{3}{4}$ pound bentonite, and $7\frac{1}{2}$ pounds water.

Result: Product not so fine or gelatinous as in test No. 1. When washed on a 200-mesh screen gave 37.5 per cent coarser than 200 mesh.

Test No. 3

Mixture: 90 per cent petroleum asphalt
10 per cent bentonite

Ground in a 12 by 12-inch ball mill for $2\frac{1}{2}$ hours with an equal weight of water. Charge consisted of $4\frac{1}{2}$ pounds asphalt, $\frac{1}{2}$ pound bentonite, and 5.0 pounds water.

Result: Product coarser and less gelatinous than in test No. 2. When washed on a 200-mesh screen gave 52.5 per cent coarser than 200 mesh.

Test No. 4

Comparative wet-grinding tests were also conducted on asphalt-bentonite and asphalt-china clay mixtures.

Mixture A: 80 per cent asphalt
20 per cent bentonite
Twice amount of water, by weight

Mixture B: 80 per cent asphalt
20 per cent china clay
Equal amount of water, by weight

Ground in a 12 by 12-inch ball mill for $2\frac{1}{2}$ hours. Samples were taken at the end of $\frac{1}{2}$, 1, $1\frac{1}{2}$, 2, and $2\frac{1}{2}$ hours. Examination of the samples, after standing for some time, showed that the bentonite mixture was in each case much finer, more homogeneous, and a superior product in every way, to the china clay mixture. The bentonite mixture after $2\frac{1}{2}$ hours grinding was an extremely fine product, whereas the china clay mixture was much coarser, not so homogeneous, and the china clay settled out to some extent on standing.

Test No. 5

Dry grinding tests on asphalt-bentonite mixtures were made in two different types of grinding machines. In the 12 by 12-inch ball mill the asphalt softened almost immediately and balled up, forming a sticky mass, and practically no grinding took place. In the laboratory type of Raymond pulverizer careful adjustment of feed and discharge resulted in a fairly fine product, but this was not nearly so fine or uniform as that made by wet grinding. The higher temperature generated in the ball mill probably accounted in a large measure for the "balling" of the asphalt.

Test No. 6

A wet-grinding test was also made on a mixture of resin and bentonite:

Mixture: 80 per cent resin
20 per cent bentonite
Twice amount of water by weight

Ground in a 12 by 12-inch ball mill for $\frac{1}{2}$ hour. The resultant product was a fine homogeneous paste from which no water separated on standing.

SUMMARY AND CONCLUSIONS

The above tests show that the addition of 20 per cent bentonite in the wet grinding of asphalt assists materially in the securing of a fine, uniform product. The paste formed with the amount of water added in test No. 1 was exceedingly smooth and homogeneous, and no separation of water took place on standing. More prolonged grinding, owing to the effecting of more perfect emulsification, might permit of the addition of even a larger amount of water for a paste of the same consistency.

Report No. 206

CONCENTRATION OF A LEAD-ZINC ORE FROM RIONDEL, B.C.

R. K. Carnochan

Shipments.—Two shipments of a lead-zinc ore were received from the Shepherd Mining Co., Riondel, B.C. Shipment No. 1 of 5 pounds was received November 10, 1923. Shipment No. 2 of 172 pounds was received December 12, 1923.

Purpose of Experimental Tests.—Tests were desired to determine a suitable method of recovering the lead and zinc values in marketable products.

Characteristics of the Ore.—The chief mineral constituents of the ore are galena, zinc blende, arsenopyrite, pyrite, calcite rhodochrosite (manganese carbonate) and quartz.

Sampling and Analysis.—Lot No. 1 was crushed to 20 mesh, a sample for analysis was cut out and ground to 100 mesh. Lot No. 2 was crushed to $\frac{1}{2}$ inch, a sample was cut out and crushed to 20 mesh. A sample for analysis was obtained from the 20-mesh sample and ground to 100 mesh. The analyses of the samples were as follows:—

	Lot No. 1	Lot No. 2
Lead.....	5.38 per cent	3.03 per cent
Zinc.....	8.58 "	4.31 "
Manganese.....	13.70 "	3.57 "
Arsenic.....	0.62 "	0.58 "
Gold.....	0.04 ounce per ton	0.01 ounce per ton
Silver.....	6.69 "	5.53 "

EXPERIMENTAL TESTS

A number of tests were conducted on shipment No. 2. These consisted of: first, tabling to remove as much lead as possible in a high-grade concentrate followed by flotation; second, selective flotation of the lead and zinc values. Although a high-grade lead product was obtained by tabling, using the first method, the recoveries were low and a satisfactory zinc product was not obtained. The second method gave the better results. A description of a few of the tests with the results obtained are given below:

Test No. 21

Procedure.—A 1,000-gramme sample was ground wet in a small ball mill for 30 minutes with 2.5 grammes of soda ash and 0.1 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal-tar creosote. The ground charge was then transferred to a small Ruth flotation machine and the lead floated by the use of 0.05 gramme sodium cyanide and 0.1 c.c. cresylic acid. The lead concentrate was re-run with 0.1 c.c. cresylic acid to clean it. The lead tailings were put back into the Ruth machine after decanting off the excess water, and the zinc floated by using 0.5 gramme of copper sulphate, 1 gramme soda ash, and 0.15 c.c. Barrett's No. 634, 0.1 c.c. TT mixture and 0.05 c.c. Aldol. The zinc concentrate was re-run to clean it.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Lead per cent	Zinc per cent	Lead	Zinc
Lead concentrate.....	5.60	44.90	9.30	69.2	10.3
Lead middling.....	7.10	12.23	11.70	23.9	16.3
Zinc concentrate.....	7.96	1.00	43.20	2.2	67.7
Zinc middling.....	5.55	0.89	1.90	1.3	2.1
Tailing.....	72.85	0.17	0.25	3.4	3.6

A screen test on the tailing gave:—

—65 + 100.....	2.9 per cent
—100 + 150 :.....	5.5 “
—150 + 200.....	12.2 “
—200.....	79.4 “

Test No. 22

Procedure.—This test is similar to test No. 21, except that different flotation reagents were used.

Reagents for flotation of lead—

Added to ball mill.....	Soda ash	1 gramme
	Water-gas tar	0.1 c.c.
	Coal-tar creosote	0.1 c.c.
Added to flotation cell.....	Sodium cyanide	0.05 gramme
	Cresylic acid.....	0.1 c.c.
To clean lead concentrate.....	Cresylic acid	0.05 c.c.

Reagents used for flotation of zinc:

Added to flotation cell.....	Copper sulphate	0.5 gramme
	Water-gas tar.....	0.1 c.c.
	TT mixture	0.15 c.c.
To clean zinc concentrate	Cresylic acid	0.05 c.c.
	Aldol	0.05 c.c.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Lead per cent	Zinc per cent	Lead	Zinc
Lead concentrate.....	8.15	33.36	8.55	84.6	13.9
Lead middling.....	9.82	2.45	10.60	7.5	20.8
Zinc concentrate.....	5.12	0.78	52.30	1.3	53.5
Zinc middling.....	4.04	1.28	9.80	1.6	7.9
Tailing.....	73.14	0.22	0.27	5.0	3.9

Remarks.—The results of the above tests indicate that the minerals were not entirely freed. The grade of the lead concentrate was low and the recoveries of both lead and zinc only fair. A number of tests were made in which the ore was ground wet to pass 200 mesh.

Test No. 27

SELECTIVE FLOTATION AT 200 MESH

Procedure.—1,000 grammes of the ore was ground for 30 minutes in a small ball mill with 0.05 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal-tar creosote. The ball mill charge was then screened wet on 200 mesh and the oversize returned to the mill and ground for the same time with the same amount of oil. The charge was again screened and the oversize reground as before. After the third grinding only a small amount of 200-mesh material remained on the screen. The charge was then dewatered and returned to the mill for mixing with the flotation reagents: 0.05 c.c. mixture of coal tar and coal-tar creosote and 2.5 grammes soda ash. The charge was then floated in a similar manner to test No. 21, except that no Aldol was used in making the zinc rougher concentrate, but 0.05 c.c. Aldol was used to clean it.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis				Per cent of values			
		Lead per cent	Zinc per cent	Gold oz/ton	Silver oz/ton	Lead	Zinc	Gold	Silver
Lead concentrate...	4.91	59.16	7.60	0.20	77.50	80.5	7.3	74.6	55.4
Lead middling....	6.11	7.52	7.20	0.02	17.82	12.7	8.6	9.3	15.8
Zinc concentrate...	6.42	0.96	55.70	0.01	8.35	1.7	69.8	2.4	7.8
Zinc middling.....	3.00	1.19	12.80	0.06	6.14	1.0	7.5	13.7	2.7
Zinc tailing.....	77.45	0.19	0.45	0.00	1.62	4.1	6.8	0.0	18.3

CONCLUSIONS

The results of the tests show that the ore is amenable to treatment by selective flotation. The above results are only indicative of what might be expected under operating conditions on a similar grade and class of ore. A 60 per cent lead concentrate should be obtained with a recovery of 85 per cent of the lead. A 55 per cent zinc concentrate should be obtained with a recovery of 75 per cent of the zinc.

In conducting the small-scale tests, sufficient concentrate was not obtained of either product to maintain the proper conditions as to pulp density for cleaning the rougher concentrates. This condition is believed to be the reason why the excessive fine grinding was necessary to obtain the results of test No. 27. To obtain reliable results, experimental tests should be conducted on a larger scale under continuous operating conditions.

The gold and silver values seem to be associated with the lead. In test No. 27, 75 per cent of the gold values and 55 per cent of the silver values reported in the lead concentrate. The low recovery of silver was no doubt due to excessive sliming.

Report No. 207

EXPERIMENTAL TESTS ON WINDPASS ORE

J. S. Godard

Shipments.—A shipment of 381 pounds of ore was received December 18, 1923 from the Windpass Mining Company, Chu Chua, B.C.

Lot No. 1 consisted of 191 pounds, and Lot No. 2 of 190 pounds.

The lots were representative of two types of ore taken from different sections of the Windpass vein.

The ore was from the Windpass properties situated in the North Thompson River valley, B.C. Access to the property is gained by Canadian National railway from Kamloops to Chu Chua station, a distance of 50 miles, thence five miles by motor-road, and the remaining four miles by pack trail.

Characteristics of the Ore.—It is reported that two distinct types of ore are found on the property. In the eastern mineralized zone there is a replacement of the brecciated country rock by massive magnetite, pyrrhotite, and chalcopyrite. Associated with these minerals are free gold, gold telluride, native bismuth, and cobaltite. Lot No. 1 is supposed to be representative of this type.

Ore from the westerly extension contains minute amounts of chalcopyrite, pyrrhotite, and pyrite scattered throughout a quartz gangue. Associated with the above minerals are grains and blades of native bismuth, free gold, and some tellurides. Lot No. 2 is supposed to be representative of this latter type.

Purpose of Experimental Tests.—An economical metallurgical treatment for the ores was desired.

Sampling and Analysis.—To obtain a sample for analysis the following procedure was carried out on both lots. The entire lot was crushed to 8 mesh and cut once using a Jones riffle sampler. One-half thus obtained was reduced to -14 mesh and cut twice. One of these portions was crushed to -48 mesh and cut to about one pound. The sample submitted for assay was cut from this latter quantity. The assays were as follows:—

	Lot No. 1	Lot No. 2
Gold, ounces per ton.....	1.65	1.35
Silver, ounce per ton.....	0.28
Copper, per cent.....	1.10	0.45
Bismuth, per cent.....	0.03	0.03
Nickel and cobalt, per cent.....	0.03	0.02

EXPERIMENTAL TESTS

In order to avoid unnecessary repetition, the following reference to procedure is given:

All samples of ore were cut by means of a Jones riffle sampler. Screen tests were made on Tyler standard screens. All grinding previous to amalgamation was done in a small porcelain laboratory ball mill using 15 pounds grey iron balls, and maintaining a pulp density of 66.7 per cent solids. Amalgamation tests of two hours' duration were carried out in a laboratory ball mill omitting the balls, with 100 grammes mercury, and a pulp density of 50 per cent solids. Separation of the mercury from the pulp was effected by means of a hydraulic classifier. Tabling was done on a laboratory size Wilfley table. Flotation tests were made in a small Ruth type flotation machine. Cyanide tests were of the usual bottle type.

Any deviation from the above methods is mentioned in the test.

A few preliminary tests were made to obtain some idea of the behaviour of the ore. Lot No. 1 was treated first, followed by lot No. 2, which in

turn was followed by tests on ore consisting of a combination of both lots in equal proportions. Some tests are omitted because of very poor results, others are included, not because of the results obtained, but to illustrate the behaviour of the ore under certain methods of treatment.

LOT NO. 1

Test No. 1—Amalgamation and Tabling of Amalgamation Tailing

Procedure.—1,000 grammes ore at -14 mesh was ground wet to 48 mesh and amalgamated. The tailing was tabled making two products, a concentrate and tailing; 45 per cent of the table tailing was used for assay purposes, and the remainder for a screen test.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Table concentrate.....	16.0	2.52	3.40	34.3	49.3
Table tailing (1).....	37.6	0.88	0.67	28.2	22.7
Table tailing (2).....	46.4	0.95	0.67	37.5	28.0

Screen Test

Mesh	Weight per cent	Au oz/ton	Per cent of values
+48	15.2	0.88	14.2
+65	12.2	0.90	10.7
+100	18.3	0.92	17.8
+150	10.1	0.82	8.7
+200	10.4	0.80	8.8
-200	33.8	1.12	39.8

Summary.—The ore is amenable to amalgamation; 28.8 per cent of the total gold was recovered by this method. The ratio of concentration by tabling was 6.5:1 and the loss during this operation was 4.63 per cent of original feed. The table concentrate was low grade and the tailing high in both gold and copper. A considerable amount of float copper passed into the tailing.

Test No. 3.—Amalgamation and Flotation of Amalgamation Tailing

Procedure.—1,000 grammes of ore at -14 mesh was ground wet to 48 mesh and amalgamated. The tailing was reground to 100 mesh and floated. About one-half of the flotation tailing was used for a screen test.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate.....	8.1	5.50	12.16	38.3	93.2
Flotation tailing (1).....	45.9	0.76	1.01	30.0	3.3
Flotation tailing (2).....	46.0	0.80	0.10	31.7	3.5

Screen Test

Mesh	Weight per cent	Au oz/ton	Per cent of values
+100	7.4	0.68	6.2
+150	15.9	0.90	17.8
+200	24.0	0.92	27.5
-200	52.7	0.74	48.5

Summary.—The recovery by amalgamation was 29.5 per cent of total gold. The ratio of concentration was 12.3: 1. The flotation concentrate containing 12.26 per cent copper with a recovery of 93 per cent as well as 38.3 per cent of the gold remaining after amalgamation, might be considered satisfactory. The flotation tailing assayed 0.10 per cent copper and 0.90 ounce gold per ton, fairly uniformly distributed as shown by the screen test.

Test No. 4.—Amalgamation followed by Cyanidation

Procedure.—1,000 grammes of the ore at -14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was dewatered and cyanided for 24 hours using 0.10 per cent solution of potassium cyanide and lime equal to 5 pounds per ton. Pulp density was 38 per cent solids. A screen test was made on the cyanide tailing.

Screen Test

Mesh	Weight per cent	Au oz/ton	Per cent of values
+ 48	3.2	1.12	8.8
+ 65	14.9	0.54	19.8
+100	20.2	0.46	22.6
+150	14.4	0.46	16.2
+200	16.6	0.36	14.6
-200	30.7	0.24	18.0

Consumption of cyanide amounted to 2.52 pounds per ton, and that of lime was 4.9 pounds per ton.

Summary.—The total recovery amounted to 75.2 per cent of the gold, of which 29.10 per cent was due to amalgamation. The results of the cyanidation were poor, due to the gold values remaining in the coarser sizes as indicated by the screen test. These results indicated that an all-slime process of cyanidation was necessary for a high extraction of the gold values.

Test No. 6.—Amalgamation, Tabling, Flotation of Table Tailing, and Cyanidation of Flotation Tailing

Procedure.—1,000 grammes ore at -14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was tabled and the table tailing reground to 100 mesh and floated. The flotation tailing was sampled and the remainder reground to -200 mesh and cyanided for 48 hours using 0.10 per cent potassium cyanide solution and lime equivalent to 6.2 pounds per ton, with a pulp density of 33 per cent solids.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Table concentrate.....	19.3	3.02	3.32	45.8	60.2
Flotation concentrate.....	7.5	3.56	4.88	21.0	34.4
Flotation tailing.....	73.2	0.58	0.08	33.2	5.4

Flotation reagents: Coal tar 40 per cent
 Coal-tar creosote 60 per cent } 0.5 pound per ton
 Pine oil No. 5 } 2 drops

Cyanide reagents: Potassium cyanide to the extent of 2.0 pounds per ton, and lime equivalent to 5.8 pounds per ton were consumed.

Summary.—The recovery by amalgamation was 22.6 per cent, which is roughly 8 per cent below the average test; 94.6 per cent of the copper was obtained in the concentrate which also contained 66.8 per cent of the gold remaining after amalgamation. The results of the cyanidation were again unsatisfactory. The cyanide heads assayed 0.58 ounces per ton, and the tailing 0.24 ounces per ton, making a recovery of 58.6 per cent. The total recovery amounted to 85.5 per cent.

Test No. 7.—Amalgamation, Flotation, and Cyanidation

Procedure.—1,000 grammes of the ore at -14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was ground to 100 mesh and floated. The flotation tailing was sampled and the remainder divided into three parts, each part being cyanided separately. The cyanide tailing was screened on 200 mesh.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate.....	13.9	3.24	6.92	37.8	92.7
Flotation tailing.....	86.1	0.86	0.09	62.2	7.3

Flotation reagents.—Same reagents used as in test No. 6.

Details of cyanidation.—

	Part A	Part B	Part C
Potassium cyanide solution, per cent.....	0.10	0.10	0.10
Lime, pounds per ton.....	10.0	7.0	7.5
Ratio ore to water.....	1 : 3.6	1 : 2.5	1 : 2.7
Time, hours.....	24	36	48
Heads, ounce per ton gold.....	0.86	0.86	0.86
Tails +200 " ".....	0.36	0.34	0.30
-200 " ".....	0.20	0.20	0.20
Recovery, per cent.....	73.2	72.2	74.4
Cyanide consumption, pounds per ton.....	2.9	2.4	3.0
Lime " ".....	7.7	5.5	6.1

Summary.—Amalgamation was responsible for a recovery of 27.9 per cent of the gold. The flotation concentrate was low in copper but

represented 92.7 per cent of the total; 37.8 per cent of the gold remaining after amalgamation was in the flotation concentrate which assayed 3.24 ounces per ton. The cyanide tailing was high in gold especially in the +200 size. Extraction of gold from -200 size showed no increase by extending the time of agitation. A slight increase in the extraction though not proportional to the increase in time was made in the +200 mesh.

Test No. 8.—Amalgamation, Flotation, and Cyanidation

Procedure.—1,021 grammes of the ore at -14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was reground to 65 mesh and floated. The flotation tailing was reground to 150 mesh and sampled. The remainder of the flotation tailing was divided into three parts and cyanided separately. Screen tests on 200 mesh were made.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate.....	9.8	4.56	9.20	38.6	89.2
Flotation tailing +200.....	21.1	0.80	0.09	14.6	1.9
Flotation tailing -200.....	69.1	0.78	0.13	46.8	8.9

Flotation reagents.—Same as in test No. 6.

Details of cyanidation:—

	Part A	Part B	Part C
Weight, grammes.....	235.3	231.8	227.2
Ratio ore to water.....	1 : 3	1 : 3	1 : 3
Cyanide solution at start.....	0.05	0.10	0.2
Lime, pounds per ton.....	6.7	6.8	7.0
Sodium peroxide, pound per ton.....	0.85	0.86	0.88
Time, hours.....	24	24	24
Cyanide consumption, pounds per ton.....	1.5	2.3	3.7
Lime " " ".....	5.5	6.0	6.1

Screen Test on Cyanide Tailing

—	Weight grammes	Weight per cent	Au oz/ton	Per cent of values	Per cent recovery
Part A +200.....	65.5	27.8	0.30	42.0	62.5
−200.....	169.8	72.2	0.16	58.0	79.4
Part B +200.....	55.7	24.1	0.28	42.4	65.2
−200.....	176.1	75.9	0.12	57.6	84.6
Part C +200.....	55.0	24.2	0.24	43.3	70.0
−200.....	172.2	75.8	0.10	56.7	87.2

Summary.—30.1 per cent of the gold was recovered by amalgamation. The flotation concentrate contained 9.20 per cent copper with a recovery of 89.2 per cent and 38.6 per cent of the gold remaining after amalgamation. It assayed 4.56 ounces per ton gold. The highest recovery by cyanidation is 87.2 per cent on -200-mesh material using 0.20 per cent solution of cyanide. Test 8C showed a total gold recovery of 94.5 per cent.

Test No. 9.—Amalgamation, Flotation, and Cyanidation

Procedure.—1,025 grammes of the -14-mesh ore was ground to 65 mesh and amalgamated. The amalgamation tailing was ground to 100 mesh with suitable flotation reagents and then floated. The flotation tailing was ground to 200 mesh and a sample equal to about one-quarter of the tailing was taken for assay purposes. The remainder was divided into three parts and each part was cyanided for 48 hours. The only variable in this test was the dilution, which was 1 : 2, 1 : 3, and 1 : 4. The cyanide solution was 0.15 per cent in each case, and lime equivalent to 8 pounds per ton was used.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	9.5	4.60	9.46	40.7	88.4
Flotation tailing.....	90.5	0.70	0.13	59.3	11.6

Flotation reagents.—Same as in test No. 6.

Details of cyanidation:—

	Part A	Part B	Part C
Weight, grammes.....	229.2	232.4	232.4
Ratio, ore to water.....	1 : 2	1 : 3	1 : 4
Cyanide per cent solution at start.....	0.15	0.15	0.15
Lime, pounds per ton at start.....	8.7	8.7	8.7
Time, hours.....	48	48	48
Cyanide consumption, pounds per ton.....	2.8	4.4	4.5
Lime " " ".....	8.5	8.0	7.5

The cyanide tailing was screened on 200 mesh and the products assayed.

Screen Test

—	Weight grammes	Weight per cent	Au oz./ton	Per cent of values	Per cent recovery
Part A +200.....	9.4	4.1	0.25	6.3	64.2
—200.....	219.8	95.9	0.16	93.7	77.1
Part B +200.....	7.9	3.4	0.22	4.6	68.5
—200.....	224.5	96.6	0.16	95.4	77.1
Part C +200.....	8.7	3.7	0.20	5.2	71.4
—200.....	224.2	96.3	0.14	94.8	80.0

Summary.—Amalgamation yielded 35.1 per cent of the gold and showed an increase over that obtained at 48 mesh. Evidently finer grinding previous to amalgamation would increase the recovery by amalgamation. Little variation is shown in the flotation from that of previous tests. The cyanide tailing averaged 0.155 ounce gold per ton. Small advantage is gained in cyanidation in a more dilute pulp. The total recovery averaged 90.5 per cent.

Amalgamation Tests.—It was thought that insufficient data had been obtained on the amalgamation phase of this work, so the following tests were made.

Each test was divided into two parts, A and B. Part A consisted of grinding wet in a ball mill 1,000 grammes of ore at -14 mesh for a certain length of time, varying in each test, then screening the ground ore and assaying the products on each screen. Part B consisted of grinding the same amount of ore the same length of time as in part A and under, as far as possible, the same conditions. The ground ore in Part B was then amalgamated using 100 grammes mercury, and a pulp density of 50 per cent solids. The mercury was separated in a hydraulic classifier, and the amalgamation tailing screened and the products assayed.

The difference in assays of each screen product before and after amalgamation based in per cent will give the amalgamation at that mesh after grinding that length of time. The time in each test number was varied, and the difference between the head samples before and after amalgamation permit the calculation of the per cent of gold amalgamated at the size that grinding that length of time would produce.

Amalgamation Tests on Lot No. 1

Test No.	PART A				PART B.				Remarks
	Product	Weight per cent	Assay Au oz/ton	Per cent of values	Weight per cent	Assay Au oz/ton	Per cent of values	Per cent head sample amalgamated	
10	-28+35	1.4	1.19	1.0	1.0	1.14	0.9	4.2	Ground 15 minutes. Head sample 1.68 oz./ton After amalgamation 1.21 oz/ton. Per cent amalgamated, 28.0.
	-35+48	5.9	1.22	4.3	6.3	1.10	5.7	9.8	
	-48+65	14.8	1.32	11.6	17.7	1.18	17.2	10.6	
	-65+100	17.2	1.40	14.3	15.6	1.18	15.3	15.6	
	-100+150	12.4	1.72	12.7	11.6	1.26	12.2	26.7	
	-150+200	15.6	1.72	16.0	15.2	1.18	14.8	31.4	
	-200	32.7	2.06	40.1	32.6	1.25	33.8	39.3	
11	-48+65	3.4	1.66	3.4	7.1	0.98	5.8	40.9	Ground 25 minutes. Head sample 1.65 oz/ton. After amalgamation 1.18 oz/ton. Per cent amalgamated, 28.5.
	-65+100	15.1	1.30	11.8	11.4	1.04	10.0	20.0	
	-100+150	14.9	1.30	11.8	14.7	1.16	14.5	10.8	
	-150+200	25.3	1.56	23.9	22.5	1.16	22.4	25.7	
	-200	41.3	1.96	49.1	44.3	1.26	47.3	35.7	
12	-65+100	9.8	1.56	9.4	6.3	0.90	5.2	42.3	Ground 35 minutes. Head sample 1.63 oz/ton After amalgamation 1.09 oz/ton. Per cent amalgamated, 33.1.
	-100+150	11.6	1.24	8.8	10.9	0.98	9.8	20.9	
	-150+200	38.0	1.59	34.9	19.0	0.98	17.0	34.7	
	-200	40.6	1.88	46.9	63.8	1.16	68.0	38.3	
13	+150	2.0	4.55	5.3	2.3	0.90	2.0	80.2	Head sample 1.71 oz/ton After amalgamation 1.03 oz/ton Ground 60 minutes. Per cent amalgamated, 39.7.
	-150+200	6.3	1.62	6.0	6.6	0.92	5.9	43.2	
	-200	91.7	1.66	88.7	91.1	1.04	92.1	37.3	

Summary.—From the above tests, it would seem that amalgamation increases with finer grinding. At 65 mesh gold begins to accumulate on the coarse screens until it reaches 4.55 ounces per ton. At 150 mesh the best recovery, 39.7 per cent of the gold, is made.

Test No. 31.—Flotation and Cyanidation

Procedure.—A sample of 783 grammes of the ore at -14 mesh was ground to 65 mesh and floated. The flotation tailing was ground to -200 mesh in a ball mill using pebbles in place of grey iron balls. About one-half the

flotation tailing was cyanided for 24 hours under the conditions as described below:—

Results of Flotation Test

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate.....	8.0	11.84	12.80	56.8	95.7
Flotation tailing.....	92.0	0.78	0.05	43.2	4.3

Flotation reagents:—

Dupont flotation oil.....	0.50 lb/ton
Pine oil No. 5.....	2 drops

Details of cyanidation:—

Flotation tailing.....	332.2 grms.
Water, c.c.....	700
Cyanide, per cent solution.....	0.2
Sodium hydroxide, pounds per ton.....	5
Heads, gold, ounce per ton.....	0.78
Tailing, gold, ounce per ton.....	0.10
Recovery per cent.....	87.2
Cyanide consumption pounds per ton.....	2.8
Sodium hydroxide consumption, pounds per ton.....	4.0

Summary.—The flotation results from this test were very satisfactory; 56.8 per cent of the gold, and 95.7 per cent of the copper were obtained in the concentrate. Cyanidation of the reground flotation tailing was satisfactory and the cyanide consumption was moderate. The total recovery was 94 per cent.

Cyanidation Tests.—A few tests were undertaken to ascertain the possibilities of direct cyanidation, that is, omitting any other metallurgical treatment previous to cyanidation. It was thought that in such a complex ore as that of lot No. 1, soluble sulphates, water soluble cyanicides, and the presence of copper would interfere with this method, tending to increase the consumption of cyanide to such an extent as to make its application practically impossible. A test for soluble sulphates was first attempted and revealed in a soluble form 0.074 gramme sulphur per 1,000 grammes of ore, a quantity that is negligible. A water soluble cyanicide test indicated that no trouble need be expected from this source. The effect of copper in the ore on cyanide was next determined by cyaniding a sample of 1,000 grammes of ore at —200 mesh for 24 hours using 0.20 per cent solution of cyanide, and a pulp density of 33 per cent solids. The copper found in 200 c.c. of this solution amounted to 0.0582 gramme, equivalent to 5.29 per cent of that present in the ore. This would amount to roughly 1.2 pounds copper per ton of ore, and necessitates, on the basis of 2 units cyanide for each cyanide soluble unit of copper, 2.4 pounds of cyanide per ton of ore for this element alone. No remedy other than that of first removing the copper by some method such as flotation, can be suggested.

Cyanidation Tests on Ore Ground to -200 mesh

Test No.	Weight grammes	Water c.c.	KCN %	NaOH lb./ton	Assay, Au oz/ton		Recovery %	Consumption lb/ton		Remarks
					Heads	Tailing		KCN	NaOH	
19	500	1,000	0.40	8	1.65	0.16	90.4	13.4	7.0	Ore crushed dry—lime used in place of caustic soda (NaOH). Time, 24 hours.
25	250	500	0.30	8	1.65	0.18	89.2	9.2	4.6	Ore ground wet, using pebbles. Time, 24 hours.
27	252	500	0.25	5	1.70	0.25	85.3	11.7	3.9	Ore crushed dry. After 21 hours cyanide solution was 0.012% cyanide and was raised to 0.20%. Time 24 hours.
28	247	500	0.25	5	1.70	0.20	88.3	12.8	Ore crushed dry. After 21 hours cyanide solution was 0.12% KCN and was raised to 0.2%. Bromo-cyanide equal to 0.1% added. Time, 24 hours
29	210	420	0.30	5	1.72	0.35	79.6	11.5	3.7	Ore crushed dry, 200 mesh. Time, 45 hours.
30	215	430	0.30	5	1.72	0.20	88.4	11.6	Ore crushed dry, 200 mesh. Cyanided for 45 hours. KCN then raised to 0.10%. Bromo-cyanide 0.04% added and agitation continued for 4 hours. Total time, 49 hours.

Procedure.—A sample of 1,000 grammes of ore at -14 mesh was ground to -48 mesh in a ball mill using pebbles in place of the usual grey iron balls, and then amalgamated with 100 grammes of mercury in a pulp density of 50 per cent solids. The mercury was separated using a hydraulic classifier, and the amalgamation tailing was ground to 200 mesh, again using pebbles. The reground amalgamation tailing was divided into four parts, one of which, No. 4, was assayed. The remaining three parts were cyanided separately, the details of which will be found tabulated below:—

	Part 1	Part 2	Part 3
Ore, grammes.....	227.3	229.0	227.0
Water, c.c.....	500	500	500
Cyanide solution, per cent.....	0.2	0.3	0.4
Caustic soda, pounds per ton.....	8	8	8
Time, hours.....	41	41	41

The results of the cyanidation tests are tabulated below:—

	Part 1	Part 2	Part 3
Heads, gold, ounces per ton	1.17	1.17	1.17
Tailings, gold, ounce per ton.....	0.33	0.28	0.2
Recovery, per cent	71.8	76.0	82.9
Cyanide consumption, pounds per ton.....	6.8	9.1	10.5
Caustic soda consumption, pounds per ton.....	4.7	3.8	4.0

Summary.—The recovery by amalgamation amounted to 29.1 per cent of the total gold. The total recoveries are:

Part 1—80 per cent.

Part 2—83 per cent.

Part 3—87.9 per cent.

The results of the cyanidation of the amalgamation tailings were disappointing, the tailings were high, and the recoveries were only fair. Considerable increase in extraction is found by using 0.40 per cent cyanide solution instead of 0.3 per cent. The difference in cyanide consumption in Parts 2 and 3 would be warranted by the increased extraction.

SUMMARY OF RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 1

1. *Amalgamation*:—

30 per cent of the gold values were recovered by amalgamation at 48 mesh.

2. Amalgamation and Flotation:—

Amalgamation at 48 mesh.....29·5 per cent of the gold values recovered.

Flotation at 48 mesh.....copper concentrate 8.1 per cent of weight of feed.

Assay of concentrate.....gold, 5.50 ounces per ton....copper, 12.26 per cent.

Recoveries in concentrate...gold, 38.2 per cent.....copper, 93.2 per cent.

Total recoveries.....gold, 56.5 per cent.....copper, 93.2 per cent.

3. Amalgamation, Flotation, and Cyanidation:—

Amalgamation at 48 mesh.....30·1 per cent of the gold values recovered.

Flotation at 65 mesh.....copper concentrate 9.8 per cent of weight of feed.

Assay of concentrate.....gold, 4.56 ounces per ton...copper, 9.2 per cent.

Recoveries in concentrate...gold, 33.6 per cent.....copper, 89.2 per cent.

Cyanidation at 200 mesh.....Heads (flotation tailing)....gold, 0.78 ounce per ton.

Cyanide tailing.....gold, 0.10 ounce per ton.

Recovery.....gold, 87·2 per cent.

Total recoveries.....gold, 94·5 per cent.....copper, 89·2 per cent.

Reagent consumption.....cyanide, 3.7 pounds per ton..lime, 6.1 pounds per ton.

4. Flotation and Cyanidation:—

Flotation at 65 mesh.....	copper concentrate, 8 per cent of weight of feed.
Assay of concentrate.....	gold, 11·84 ounces per ton...copper, 12·8 per cent.
Recoveries in concentrate.....	gold, 56·8 per cent.....copper, 95·7 per cent.
Cyanidation at 200 mesh:—	
Heads (flotation tailing).....	gold, 0·78 ounce per ton.
Cyanide tailing.....	gold, 0·1 ounce per ton.
Recovery.....	gold, 87·2 per cent.
Total recoveries.....	gold, 94 per cent.....copper, 95·7 per cent.
Reagent consumption.....	cyanide, 2·8 pounds per ton..caustic soda, 4 pounds per ton.

5. Straight Cyanidation at 200 mesh:—

Heads, gold.....	1·65 ounces per ton.
Cyanide tailing, gold.....	0·18 ounce per ton.
Recovery of gold values.....	89·2 per cent.
Reagent consumption.....	cyanide, 9·2 pounds per ton..caustic soda, 4·6 pounds per ton.

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 1

1. Thirty per cent of the gold values are recovered by amalgamation. Transportation difficulties would give favour to this method as the first step in the treatment of the ore.

2. Very favourable results were obtained on the recovery of the copper values by flotation following amalgamation at 48 mesh. Only 56·5 per cent of the gold values are recovered, showing that this method must be supplemented for a further recovery of the gold in the ore. A high-grade copper concentrate is produced assaying 12·26 per cent copper and 5·50 ounces gold per ton.

3. Very favourable results were obtained by amalgamation followed by flotation and cyanidation of the flotation tailing. The flotation concentrate produced was of lower grade, but raising this is only a matter of manipulation. The final cyanide tailing assayed 0·1 ounce per ton in gold, showing a total recovery of 94·5 per cent of the gold values. The recovery of the copper values was 89·2 per cent.

4. Very favourable results were also obtained by flotation at 65 mesh, followed by cyanidation at 200 mesh. The flotation concentrate was very high grade, the gold recoverable by amalgamation reporting in the concentrate. The assay of the flotation concentrate showed 11·84 ounces gold per ton, and 12·8 per cent copper. The final cyanide tailing assayed 0·1 ounce gold per ton. The total recoveries were 94 per cent of the gold values and 95·7 per cent of the copper values. As far as the metallurgy of the ore is concerned, this method shows that amalgamation is not a necessary part of the flow-sheet. The use of this method would depend on transportation facilities to the smelters or some method of treatment for the copper concentrate at the mine. Should flotation be adopted good results can be obtained by using coal tar and coal-tar creosote mixed in the proportions of 40 and 60 per cent respectively. Pine oil No. 5 is a suitable frothing reagent as the pulp is neutral. The best flotation results are obtained on the ore ground to between 48 and 65 mesh.

5. Straight cyanidation shows a recovery of 90 per cent of the gold values. The chief objection is the high consumption of cyanide. The removal of the copper by flotation reduces the consumption from 9 pounds to 2·8 pounds per ton of ore treated. Fine grinding is essential for a good recovery of the gold values by cyanidation. Cyanidation of the amalga-

mation tailing shows no increased total recovery over straight cyanidation. The last 0.2 ounce per ton of remaining values is very difficult to extract.

6. The introduction of table concentration into the flow-sheet is not necessary. Besides complicating the flow-sheet no increased recoveries are shown.

LOT NO. 2

Amalgamation.—A series of amalgamation tests were conducted following a similar procedure to those on lot No. 1. Tabulated below are the results obtained:—

Test No. 3 is a screen analysis of the -14 mesh ore ground for 20 minutes in a small ball mill.

Test No. 4 is a screen analysis of the amalgamation tailing from a similar sample of the -14 mesh ore ground for 20 minutes and amalgamated.

Tests Nos. 5 and 6, 7 and 8, and 9 and 10, are similar tests, grinding for 30, 45, and 60 minutes respectively.

Summary of Results.—The percentage of gold values amalgamated increased with finer grinding until on ore ground to 100 mesh, 51 per cent was amalgamated. As in lot No. 1, gold began to accumulate when the ore was ground to 65 mesh until it reached 3.88 ounces per ton when ground to 100 mesh.

Amalgamation Tests on Lot No. 2

Test No.	Product	Weight %	Au oz./ton	Per cent of values	Test No.	Product	Weight %	Au oz./ton	Per cent of values	Per cent head sample amalgamated	Remarks
3	+ 48	4.7	0.86	3.0	4	+ 48	1.9	0.80	1.9	7.0	Head sample..... 1.35 oz./ton, gold
	- 48+65	14.6	1.08	11.7		- 48+65	11.5	0.78	11.2	27.7	After amalgamation.. 0.80 oz./ton, gold
	- 65+100	19.3	1.10	15.7		- 65+100	22.9	0.80	22.8	27.3	Per cent amalgamated 40.4
	- 100+150	13.3	1.22	12.0		- 100+150	15.2	0.74	14.1	39.3	
	- 150+200	16.5	1.40	17.2		- 150+200	9.2	0.70	8.0	50.0	
	- 200	31.6	1.72	40.4		- 200	39.2	0.86	42.0	50.0	
5	+ 65	4.1	1.84	5.2	6	+ 65	3.2	0.70	2.7	62.0	Head sample..... 1.45 oz./ton, gold
	- 65+100	14.5	1.34	13.4		- 65+100	14.0	0.74	12.7	44.8	After amalgamation.. 0.81 oz./ton, gold
	- 100+150	17.6	1.26	15.3		- 100+150	16.1	0.74	14.6	41.2	Per cent amalgamated 44.1
	- 150+200	24.0	1.26	21.0		- 150+200	29.0	0.74	26.3	41.2	
	- 200	39.8	1.66	45.1		- 200	37.7	0.94	43.5	43.4	
7	+ 100	5.3	2.94	11.1	8	+ 100	6.7	0.68	6.2	76.9	Head sample..... 1.42 oz./ton, gold
	- 100+150	11.1	1.20	9.4		- 100+150	9.3	0.66	8.2	45.0	After amalgamation.. 0.74 oz./ton, gold
	- 150+200	16.8	1.08	12.7		- 150+200	26.4	0.62	22.1	42.6	Per cent amalgamated 47.8
	- 200	66.8	1.40	66.8		- 200	57.6	0.82	63.5	41.4	
9	+ 100	2.1	3.88	6.0	10	+ 100	1.3	0.72	1.4	81.4	Head sample..... 1.35 oz./ton, gold
	- 100+150	9.5	1.54	10.8		- 100+150	3.0	0.58	2.6	62.2	After amalgamation.. 0.86 oz./ton, gold
	- 150+200	8.4	1.02	6.3		- 150+200	14.0	0.56	11.8	45.0	Per cent amalgamated 51.0
	- 200	80.0	1.30	76.9		- 200	81.7	0.68	84.3	47.6	

Amalgamation, Flotation, and Cyanidation

Procedure.—A sample of 1,000 grammes was ground to 65 mesh and amalgamated. The amalgamation tailing was floated. The flotation tailing was sampled, screened, and the remainder reground to -200 mesh, divided into two parts, and each part cyanided separately for 24 hours.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate.....	5.1	2.76	6.40	19.6	85.2
Flotation tailing.....	94.9	0.61	0.06	80.4	14.8

Flotation reagents: Water-gas tar 50 per cent.... } 0.50 pound per ton.
 Coal-tar creosote 50 per cent }
 Pine oil, No. 5..... 2 drops.

Result of screen test on 317 grammes flotation tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Cu %	Au oz/ton	Cu	Au
-65+100.....	14.3	0.64	0.04	14.8	10.2
+150.....	22.1	0.64	0.05	23.3	18.6
+200.....	13.4	0.60	0.05	13.0	11.9
-200.....	50.2	0.60	0.07	48.9	59.3

Details of cyanidation:—

	Part 1	Part 2
Ore—grms.....	310	304.2
Water, c.c.....	600	600
Potassium cyanide, per cent solution.....	0.10	0.20
Lime, pounds per ton.....	6.8	6.8

Results of cyanidation:—

Heads, gold, ounces per ton.....	0.61	0.61
Tailing, gold, ounces per ton.....	0.06	0.06
Recovery, per cent.....	91.8	91.8
Cyanide consumption, pounds per ton.....	3.0	5.2
Lime consumption, pounds per ton.....	6.2	6.11

Summary.—The head sample assayed 1.35 ounces per ton and the tailing 0.06 ounce per ton, making a total recovery of 95.5 per cent of the gold values. Amalgamation yielded 46.6 per cent. The flotation concentrate is low both in grade and recovery of gold. A fair recovery of the copper values was made. Cyanidation results on this test are above the average.

Test No. 23.—Amalgamation followed by Cyanidation

Procedure.—1,000 grammes of the ore at -14 mesh was ground to 35 mesh and amalgamated. The amalgamation tailing was ground to 200 mesh, sampled, and divided into three parts, each of which was cyanided separately.

Details of cyanidation:—

	Part 1	Part 2	Part 3
Ore, grms.....	239.7	243.7	234.4
Water, c.c.....	500	500	500
Potassium cyanide, per cent solution.....	0.11	0.2	0.3
Caustic soda, pounds per ton.....	8.0	8.0	8.0
Time, hours.....	24	24	24

Results of cyanidation:—

Head sample, gold, ounce per ton.....	0.86	0.86	0.86
Tailing, gold, ounce per ton.....	0.15	0.12	0.13
Recovery, per cent.....	82.5	86.1	84.4
Cyanide consumption, pounds per ton.....	3.1	2.5	3.1
Caustic soda consumption, pounds per ton.....	7.0	4.3	4.3

Summary.—Amalgamation yielded 36 per cent of the total gold. The total recovery averaged 90.5 per cent. Little advantage is gained by using stronger cyanide solution.

Straight Cyanidation.—Previous to attempting the direct cyanidation of ore of this lot, several tests similar to those conducted on lot No. 1 were made.

A test for soluble sulphates showed only small amounts present in the ore. A water soluble cyanide test using 0.30 per cent solution of potassium cyanide showed that 1.2 pounds per ton cyanide would be consumed in this manner. Cyanidation in 0.20 per cent solution cyanide showed that 1.53 per cent of the copper present was soluble in a cyanide solution of this strength. This would account for the consumption of 0.28 pound per ton cyanide on the basis that one unit copper would require two units of cyanide.

Test No. 14—Cyanidation of Dry Ground Ore

Procedure.—A 500-gramme sample was crushed to 150 mesh and divided into two equal parts. One part was screened on 200 mesh, the other was cyanided for 24 hours using 0.1 per cent solution of cyanide, and lime equivalent to 8 pounds per ton.

—	Mesh	Weight %	Au oz/ton	Per cent of values	Recovery per cent
Head sample.....	+200	15.1	1.28	13.7
	—200	84.9	1.42	86.3
Tailing.....	+200	13.6	0.22	30.3	82.8
	—200	86.4	0.08	69.7	94.4

Cyanide consumption, 1.2 pounds per ton. Lime consumption, 7.3 pounds per ton.

Summary.—The average recovery was 92.8 per cent. Finer grinding is essential to the production of a low tailing.

Test No. 24.—Cyanidation of Wet Ground Ore

Procedure.—A 500-gramme sample at —14 mesh was ground in a ball mill, using pebbles, to —200 mesh, and cyanided for 24 hours; 0.1 per cent solution cyanide, and caustic soda equal to 6 pounds per ton used. Pulp density 50 per cent solids. The cyanide tailing was filtered and washed, and a sample equal to about one-half assayed. The remainder was re-cyanided for 24 hours using 0.1 per cent solution cyanide and caustic soda equal to 4 pounds per ton. Pulp density 25 per cent solids.

Results:—

First cyanidation:—	
Heads, gold, ounces per ton.....	1.35
Tailing, gold, ounce per ton.....	0.09
Recovery, per cent.....	93.3
Cyanide consumption, pounds per ton.....	1.3
Caustic soda consumption, pounds per ton.....	4.4
Second cyanidation:—	
Heads, gold, ounce per ton.....	0.09
Tailing, gold, ounce per ton.....	0.05
Recovery, per cent.....	44.5
Cyanide consumption, pounds per ton.....	0.13
Caustic soda consumption, pounds per ton.....	2.4
Totals:—	
Cyanide consumption, pounds per ton.....	1.43
Caustic soda consumption, pounds per ton.....	6.8

Summary.—The total recovery amounted to 96.3 per cent, of which 93.3 per cent was due to the first cyanidation. From this it hardly seems necessary to re-cyanide the tailing from the first operation.

SUMMARY OF RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 2

1. *Amalgamation:—*

At 35 mesh, 36 per cent of the gold is recovered.
“ 48 “ 40 “ “ “ “
“ 65 “ 45 “ “ “ “
“ 100 “ 51 “ “ “ “

2. *Amalgamation, Flotation, and Cyanidation:—*

Amalgamation at 65 mesh.....	46.6 per cent of the gold recovered.
Flotation at 65 mesh.....	Copper concentrate, 5.1 per cent weight of feed.
	Assay concentrate, gold, ounces per ton, 2.76;
	copper, 6.40 per cent.
	Recovery in concentrate, gold, 19.6 per cent;
	copper, 85.2 per cent.
Cyanidation at 200 mesh.....	Head (flotation tailing), gold, 0.61 ounce per ton.
	Tailing, gold, 0.06 ounce per ton.
	Recovery, gold, 91.8 per cent.
Total recoveries gold, 95.5 per cent; copper, 85.2 per cent.	
Reagent consumption: cyanide, 3 pounds per ton; lime, 6 pounds per ton.	

3. *Amalgamation and Cyanidation:—*

Amalgamation at 35 mesh.....	36 per cent of the gold recovered.
Cyanidation at 200 mesh.....	Head (amalg. tailing), gold, 0.86 ounce per ton.
	Tailing, gold, 0.13 ounce per ton.
	Recovery, gold, 85 per cent.
Total recoveries: gold, 90.5 per cent.	
Reagent consumption: cyanide, 2.5 to 3 pounds per ton; caustic soda, 4.3 pounds per ton.	

4. *Straight Cyanidation at 200 mesh:—*

Assay of heads.....	Gold, 1.35 ounces per ton.
Assay of tailing.....	Gold, 0.09 ounce per ton.
Recovery.....	Gold, 93.3 per cent.
Reagent consumption.....	Cyanide, 1.3 ounces per ton.
	Caustic soda, 4.4 pounds per ton.

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 2

1. Amalgamation showed a slightly increased recovery of the gold values as compared with lot No. 1.

2. The flotation concentrate was lower grade due to the smaller copper content as compared with lot No. 1. The grade could be increased by cleaning operations. By amalgamation, flotation, and cyanidation, a total recovery of 95 per cent of the gold values, and 85 per cent of the copper values was obtained.

3. Amalgamation and cyanidation gave a recovery of 90 per cent of the gold values. This should have showed as good results as straight cyanidation.

4. The ore of this lot is comparatively clean as compared with lot No. 1. Straight cyanidation gave a recovery of 93 per cent of the gold values. The cyanide and lime consumption was moderate. This class of ore could very probably be treated successfully by this method.

EXPERIMENTAL TESTS ON LOTS NOS. 1 AND 2 COMBINED

Several tests were made on ore consisting of equal parts of lots Nos. 1 and 2. These tests are lettered alphabetically and the results of those which showed any merit are given.

Test B—Amalgamation, Flotation, and Cyanidation

Procedure.—500 grammes of each lot of ore at -14 mesh were cut in a Jones riffle sampler. The two were combined and ground to 65 mesh and amalgamated. The amalgamation tailing was floated. The flotation tailing was sampled, screened, and assayed, and the remainder ground to -200 mesh, then divided into two parts, each part being cyanided separately.

Results.—Amalgamation yielded 37.3 per cent of the total gold contained in the head sample.

Flotation:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate.....	5.8	5.60	11.0	34.4	90.8
Flotation tailing.....	94.2	0.71	0.07	65.6	9.2

Analysis of flotation concentrate:—

Gold.....	5.60 ounces per ton	Iron.....	24.30 per cent
Copper.....	11.00 per cent	Sulphur.....	21.46 "
Nickel and cobalt...	0.17 "	Tellurium.....	trace
Bismuth.....	1.40 "	Insoluble.....	29.35 per cent

Reagents:—

Water-gas tar 50 per cent and coal-tar creosote 50 per cent....	0.50 pound per ton.
Pine oil No. 5.....	2 drops.
Pulp density in flotation cell—33 per cent solids.	

Screen test on 270 grammes of flotation tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
+100.....	21.1	0.78	0.05	23.3	15.9
+150.....	18.3	0.78	0.05	20.2	13.9
+200.....	10.6	0.75	0.06	11.3	9.7
-200.....	50.0	0.64	0.08	45.2	60.5

Details of cyanidation:—

	Part 1	Part 2
Ore, grms.....	322	324
Water, c.c.....	600	600
Cyanide solution, per cent.....	0.10	0.15
Time, hours.....	24	24
Cyanide consumption, pounds per ton.....	3.2	4.9
Lime consumption, pounds per ton.....	6.0	6.1
Assay of tailing, gold, ounce per ton.....	0.1	0.1
Per cent of total gold cyanided.....	35.3	35.3
Recovery from flotation tailing.....	86%	86%

Summary.—The gold recoveries may be summarized as follows:—

By amalgamation.....	37.3 per cent
By flotation.....	21.6 “
By cyanidation.....	35.3 “

In cyaniding the flotation tailing no difference is found in extraction between the 0.1 per cent and 0.15 per cent cyanide solutions.

Test C.—Cyanidation of Slimed Ore

Procedure.—1,000 grammes of ore, 500 grammes from each lot, at –14 mesh was ground to –200 mesh in a ball mill. One-quarter of the pulp was used for head sample and the remainder cut into three parts, each part being cyanided separately. Head sample assayed 1.48 ounces per ton, gold.

	Part 1	Part 2	Part 3
Ore, grms.....	248	252	244
Water, c.c.....	500	500	500
Cyanide, per cent solution.....	0.1	0.2	0.3
Time, hours.....	24	24	24
Cyanide consumption, pounds per ton.....	4.0	5.2	5.5
Lime consumption, pounds per ton.....	7.2	6.9	6.9
Assay of tailings, ounce per ton.....	0.48	0.10	0.10
Recovery per cent.....	67.5	93.4	93.4

Summary.—All the cyanide in Part 1 was consumed. This would account for the low recovery. A recovery of 93.4 per cent of the gold values was obtained using 0.2 per cent cyanide solution. The stronger solution used in Part 3 produced no better results.

Test D.—Flotation and Cyanidation

Procedure.—1,000 grammes of ore, 500 grammes of each lot, at –14 mesh was ground to 48 mesh with suitable flotation reagents and floated in a Ruth machine. The flotation concentrate was cleaned and the tailing ground to –200 mesh. About two-thirds of the tailing was used for sampling and assaying and the remainder cyanided for 24 hours using 0.3 per cent cyanide solution and caustic soda equal to 14 pounds per ton.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate.....	0.3	32.77	28.08	7.2	13.0
Flotation middling.....	49.2	9.04	5.83	31.3	42.6
Flotation tailing.....	95.0	0.88	0.30	61.5	44.4

Reagents:—

Coal tar.....0.50 pounds per ton
Fumol, No. 6.....2 drops

No further reagents added during clean-up.

Details of cyanidation:—

Ore, grms.....	320
Water, c.c.....	600
Cyanide, per cent solution.....	0.3
Caustic soda, pounds per ton.....	14.0
Cyanide consumption, pounds per ton.....	6.5
Caustic soda consumption, pounds per ton.....	5.7
Heads, gold, ounce per ton.....	0.88
Tailing, gold, ounce per ton.....	0.10
Recovery, per cent.....	88.7

An addition of cyanide was made at 19 hours.

Summary.—Although the assay values of the flotation concentrate were very high, the quantity was so small that the recovery was only 7.2 per cent of the gold and 13.0 per cent of the copper. The middlings carried fair values and the tailing was high both in gold and copper. The distribution of the gold is:—

In flotation concentrate.....	7.2 per cent
In flotation middling.....	31.3 “
In cyanide solution.....	54.5 “
In cyanide tailing.....	7.0 “

In addition to the above the flotation concentrate carried 6.59 ounces per ton silver.

Test E.—Flotation and Cyanidation

Procedure.—1,000 grammes ore, 500 grammes of each lot, at -14 mesh was ground to -48 mesh in a ball mill with flotation reagents and floated in a Ruth machine. The flotation tailing was ground to -200 mesh and about two-thirds used for assaying, the remainder being cyanided for 24 hours.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate.....	2.1	21.94	22.08	31.6	73.4
Flotation middling.....	12.0	2.56	0.90	21.1	17.1
Flotation tailing.....	85.9	0.80	0.07	47.3	9.5

Reagents:—

P.T. and T. Co., No. 400.....0.50 pounds per ton
Pine oil, No. 350.....2 drops

No further reagents used for clean-up

Details of cyanidation:—

Ore, grms.....	282
Water, c.c.....	600
Cyanide, per cent solution.....	0.1
Caustic soda, pounds per ton.....	8.6

Heads, gold, ounce per ton.....	0.8
Tailing, gold, ounce per ton.....	0.14
Recovery, per cent.....	82.5
Cyanide consumption, pounds per ton.....	5.5
Caustic soda consumption, pounds per ton.....	4.4

A sample of the cyanide solution was tested after 18 hours and found to contain no free cyanide. Cyanide was added to bring the strength up to 0.15 per cent.

Summary.—The flotation concentrate contained 31.6 per cent of the gold and 73.4 per cent of the copper and showed a marked improvement over test D. A more satisfactory copper tailing was also obtained. Cyanidation was not so effective as in test D, due to the absence of free cyanide when first tested.

The head sample assayed 1.45 ounces gold per ton, and the cyanide tailing 0.14 ounce per ton, showing a total recovery of 90.4 per cent of the gold values.

Test G.—Amalgamation and Cyanidation

Procedure.—1,000 grammes ore, 500 grammes of each lot, at -14 mesh was ground wet to 100 mesh and amalgamated in a 1:1 pulp for 2 hours using 100 grammes mercury. The amalgamation tailing was re-ground to -200 mesh and a sample equal to about one-quarter of the total weight removed. The remainder was divided into three parts, each part being cyanided separately.

Cyanidation Results:—

	Part 1	Part 2	Part 3
Ore, grms.	244.2	240.2	244.0
Water, c.c.	500	500	500
Caustic soda, pounds per ton.....	8	8	8
Cyanide, per cent solution.....	0.1	0.2	0.25
Heads, gold, ounce per ton.....	0.83	0.83	0.83
Tailing, gold, ounce per ton.....	0.14	0.14	0.15
Recovery, per cent.....	83.2	83.2	81.9
Cyanide consumption, pounds per ton.....	2.0	3.2	3.5
Caustic soda consumption, pounds per ton....	3.3	3.2	3.1

Summary.—The total recoveries were as follows: Part 1—90.6 per cent; Part 2—90.6 per cent; Part 3—90.0 per cent. Amalgamation tailing assayed 0.83 ounces per ton making a recovery of 44.6 per cent by amalgamation.

SUMMARY OF EXPERIMENTAL TESTS ON LOTS 1 AND 2 COMBINED

1. Amalgamation, Flotation, and Cyanidation:—

Amalgamation at 65 mesh.....	37.3 per cent of gold values recovered.
Flotation at 65 mesh.....	Copper concentrate—5.8 per cent of feed.
	Assay concentrate, gold, 5.6 ounces per ton, copper 11.0 per cent.
	Recovery, gold 34.4 per cent, copper 90.8 per cent.
Cyanidation at 200 mesh.....	Heads (flotation tailing), gold, 0.71 ounce per ton
	Cyanide tailing, gold, 0.10 ounce per ton.
	Recovery, gold 86 per cent.
Total recoveries.....	Gold, 94.2 per cent; copper, 90.8 per cent.
Reagent consumption.....	Cyanide, 3.2 pounds per ton; lime, 6 pounds per ton.

2. Flotation and Cyanidation:—

Flotation at -48 mesh.....	Copper concentrate, 2.1 per cent of feed. Assay of concentrate, gold, 21.94 ounces per ton, copper 22.08 per cent. Recovery in concentrate, gold, 31.6 per cent, cop- per 73.4 per cent.
Cyanidation at 200 mesh.....	Heads (flotation tailing), gold, 0.80 ounce per ton. Cyanide tailing, gold, 0.14 ounce per ton. Recovery, gold, 82.5 per cent.
Total recoveries.....	Gold, 90.4 per cent; copper, 73.4 per cent.
Reagent consumption.....	Cyanide, 5.5 pounds per ton; caustic soda, 4.4 pounds per ton.

3. Amalgamation and Cyanidation:—

Amalgamation at 100 mesh.....	44.6 per cent of gold values recovered.
Cyanidation at 200 mesh.....	Heads (amalgamation tailing), gold, 0.83 ounce per ton. Cyanide tailing, gold, 0.14 ounce per ton. Recovery, gold, 83.2 per cent.
Total recovery of gold values.....	90.6 per cent.
Reagent consumption.....	Cyanide, 2 to 3.5 pounds per ton. Caustic soda, 8.3 pounds per ton.

4. Straight Cyanidation at 200 mesh:—

Heads, gold, ounces per ton.....	1.48
Cyanide tailing, gold, ounce per ton.....	0.10
Recovery per cent, gold ounces per ton.....	93.4
Reagent consumption....Cyanide, pounds per ton.....	5.2
Lime, pounds per ton.....	6.9

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS ON LOTS 1 AND 2 COMBINED

1. Similar results were obtained by amalgamation as were obtained on the separate lots.

2. Flotation at 48 mesh gave good results. By recleaning the rougher concentrate, a copper concentrate was obtained representing 2.1 per cent of the feed and assaying 22 ounces per ton gold and 22 per cent copper. In the test work on lot No. 1 the rougher concentrate was not recleaned. Flotation and cyanidation gave a recovery of 90 per cent of the gold values.

3. Amalgamation and cyanidation showed a recovery of 90 per cent of the gold values.

4. Straight cyanidation gave a good recovery of the gold values but with a high consumption of cyanide and lime.

GENERAL CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS

1. The ores represented by the sample lots submitted can be treated separately or mixed together. If mixed, the flow-sheet of the more refractory ore as represented by lot No. 1 will have to be followed.

2. Thirty per cent of the gold values can be recovered by amalgamation at 35 mesh. The adoption of amalgamation will depend on whether it is more economical to recover this percentage of gold or allow it to report in the flotation concentrate.

3. Flotation is essential to remove the refractory minerals in the ore of lot No. 1 before cyaniding, otherwise the consumption of cyanide will be excessive and prohibitive. A high-grade flotation concentrate can be obtained which will have to be shipped to the smelter or treated at the

mine for the recovery of the copper and gold values. This concentrate will amount to 2 to 4 per cent of the feed and assay 20 ounces gold and 20 per cent copper if amalgamation is not included in the flow-sheet, and 10 ounces gold and 20 per cent copper if amalgamation is included. Flotation between 48 and 65 mesh gives the best results.

4. Cyanidation is essential for a high recovery of the gold values. An all-slime process, grinding to 200 mesh is necessary for a high extraction.

5. The ore of lot No. 2 being comparatively clean and having a low copper content, good results were obtained by straight cyanidation with a moderate consumption of cyanide. It might be feasible to grind this ore separately and allow it to enter the cyanide circuit of the ore from lot No. 1.

6. The results of the experimental tests showed recoveries of both the gold and copper values above 90 per cent.

Report No. 208

THE CONCENTRATION OF GARNETIFEROUS GNEISS FROM BARRY'S BAY, ONTARIO

R. K. Carnochan

Shipment.—Three lots of garnetiferous gneiss were received at the Ore Dressing and Metallurgical Laboratories, as follows:—

Lot No. 1	received	December	5, 1923	—gross weight	40 pounds.
" 2	"	"	13, 1923	—	591 "
" 3	"	January	4, 1924	—	3,000 "

The shipments were submitted by the Bancroft Mines Syndicate Ltd., 18 Toronto Street, Toronto, Ont., from the company's property in the vicinity of Barry's Bay, Ont.

Purpose of Experimental Tests.—The company had decided to erect a new concentrator to replace the one recently destroyed by fire. Markets had been built up for their product and in order to retain these it was necessary to lose no time in re-commencing operations. The old method of concentration, which consisted of crushing to 6 mesh, sizing, and tabling, did not give a product high enough in garnet content to meet the demands of the trade. The company requested the assistance of the Department in determining a method of concentration that would produce a concentrate suitable for the trade they had already built up.

Conditions Governing the Method of Concentration to be Applied.—The methods to be applied to the concentration of the garnet rock of these particular deposits were governed by the following conditions:

The trade required a product of 90 per cent garnet content consisting of 6-mesh material with very little fines. The manufacturers desired to finish the crude product to meet their own requirements for the manufactured article.

The absence of an available supply of sufficient water in close proximity to the mill site for wet concentration; the advantages of operating a dry mill during the winter months; and the cost of drying wet-concentration products were factors in favour of a dry-concentration process, provided that a high-grade product could be obtained by this means. The com-

paratively small tonnage to be treated, not more than 30 tons of rock per day; the high garnet content of the rock, approximately 40 per cent garnet; the abundant supply of rock of this grade, considering the tonnage to be treated, were important factors in favour of a simple process. The tonnage to be milled was not large enough to permit of the erection of an elaborate plant to obtain extremely high recoveries of the garnet content, and too large a percentage of fines was not acceptable to the trade. The high garnet content of the rock and the abundant supply of such rock permitted a waste of considerable garnet in the tailing, especially of fines that could not be sold. If it were found desirable and economical to improve recoveries, regrinding mills and further concentration units could be added as conditions demanded, and would depend to a large extent on the sale of a larger proportion of fines.

Characteristics of the Garnet Rock.—The rock containing the garnets is a gneiss, and consists of about 40 per cent garnet mineral with large amounts of black mica and hornblende. The garnets in the deposits being worked at present range in size from about $\frac{3}{8}$ inch to minute crystals. Garnets of an inch and larger are found in certain of the deposits. The garnets are of a deep red colour and have been found to be very suitable for abrasive purposes.

EXPERIMENTAL TESTS

Lot No. 1.—Wet Concentration by Jigging and Tabling

This lot was crushed to pass a 6-mesh Tyler standard screen, and sized on 8-, 10-, 14-, and 20-mesh standard Tyler screens, and on 40- and 100-mesh screens. The sizes coarser than 20 mesh were jigged on a laboratory Richards pulsating-jig and the finer sizes tabled on a laboratory Wilfley table.

A concentrate was produced representing 35 per cent of the rock treated. A clean tailing was obtained. The concentrates, both jig and table, contained some black hornblende, but would average 90 per cent garnet. The test indicated that this method of concentration could be applied successfully to the recovery of the garnet of this particular sample.

Lot No. 2.—Dry Concentration on a Sutton, Steele and Steele Table

This lot was crushed to 6 mesh and sized on 8-, 10-, 14-, 20-, 28-, 35-, 48-, 65-, and 100-mesh Tyler standard screens. Concentration tests were made on all the sizes with the exception of the — 100 mesh, on a Sutton, Steele and Steele table. Various adjustments were made to the table, such as the cloth used for the table top, the distribution and volume of the wind-pressure, slope of the table, speed and stroke of the table, rate of feed, etc. These tests showed that a separation of the micaceous gangue could be made, but the garnet concentrate contained a large percentage of black hornblende. It was decided that dry concentration would not be successful on this class of garnet rock on account of the large amount of hornblende present, which is too nearly of the same specific gravity as the garnet to permit of a close separation by this means.

Lot No. 2.—Wet Concentration on a Wilfley Table

The sized material of the previous test run on the dry table was concentrated on a standard Wilfley table. No difficulty was experienced in

obtaining a good separation. The garnet concentrate obtained represented 51.8 per cent of the feed to the table. It contained an appreciable amount of black hornblende, but would contain approximately 90 per cent garnet. The tailing was clean. The good separation, especially on the coarser sizes, depended on the close sizing of the feed and the manipulation of the table. The grade of the rock treated (50 per cent garnet) was a factor in obtaining such a good separation.

Lot No. 3.—Wet Concentration by Jigging Unsized Material

This lot was crushed to pass a 6-mesh Tyler standard screen. An attempt was made to concentrate the —6-mesh material, without sizing, in a two-compartment James jig. It was found that a good concentrate and hutch were produced by the first jig, but a very poor concentrate and hutch were obtained from the second jig if a clean tailing were made. The test indicated the possibility of treating unsized material by jigging with the subsequent regrinding and jigging of a middling product from some of the compartments, and the tabling of the hutch products of these compartments.

Lot No. 3.—Wet Concentration by Jigging and Tabling Sized Material

The products from the previous test were dried and mixed with the remainder of the lot. The lot was then sized on 8-, 10-, 14-, 20-, 28-, and 35-mesh Tyler standard screens. The sizes coarser than 20 mesh were concentrated in a two-compartment James jig and the finer sizes on a standard Wilfley table. This lot was not so high grade as the previous ones and contained a larger amount of hornblende. Although a good separation was made with a comparatively clean tailing, the concentrate was not so high grade as in the previous small-scale tests that had been made on the higher grade rock. The concentrate represented 32 per cent of the material treated.

SUMMARY OF EXPERIMENTAL TESTS

The three lots submitted varied in garnet content, in the amounts of hornblende present, and consequently in their amenability to concentration.

Dry concentration was not successful. On the higher grade lot on which it was tried, a satisfactory concentrate could not be obtained due to the hornblende being of almost the same specific gravity as the garnet.

Lots Nos. 1 and 2 concentrated very nicely by close sizing, jigging or tabling the various sizes. Lot No. 3 being of lower grade material, the concentrates produced by the same methods of treatment, were not so high grade nor the tailing so clean as those from the other lots.

CONCLUSIONS

The results of the test work showed three simple methods of wet concentration that should be given due consideration in producing a suitable product for the trade. All three gave good results and can be tried out at the mine in more detail and on a much larger scale to prove definitely which is the most economical.

1. *Table Concentration of the Sized Material.* The success of this method will depend on very close sizing of the rock crushed to 6 mesh, and

on the careful manipulation of the tables in concentrating the coarser sizes.

2. Jig and Table Concentration of the Sized Material. Jigs, as a general rule, have been found to be more adaptable than tables to concentration of material coarser than 20 mesh. In most cases they will give a cleaner product, have a greater capacity, do not require as closely graded material, and are more easily manipulated. Tabling does better work on the finer sizes.

3. Jig and Table Concentration of the Unsized Material. It may be found more economical to jig the unsized material, thus eliminating the cost of sizing. By using jigs with several compartments, a clean concentrate and hutch could be obtained from the first two compartments, a middling product and hutch middling from the remaining compartments, and a clean tailing. The middling product could be reground in wet rolls and returned to the jig circuit, and the hutch middling concentrated on tables.

Report No. 209

GOLD ORE FROM MINE CENTRE, ONTARIO

C. S. Parsons

Shipment.—A shipment of 200 pounds of gold ore was received January 4, 1924, from J. S. Hillyer, Esq., 4 Chester Terrace, Duluth, Minn., U.S.A.

Location of Deposit.—The sample was taken from the property known as the Lucky Coon mine, situated on mining location P-655, near Mine Centre, Rainy River district, Ont.

Characteristics of Sample.—The sample is a free milling gold quartz ore, containing a small amount of iron pyrite and is supposed to be representative of what would be considered the milling ore from the mine.

Sampling and Analysis.—The 200-pound sample was crushed to 8 mesh and cut in a riffle sampler to 100 pounds. The 100 pounds was further reduced to 14 mesh. The sample for assay was obtained from the 14-mesh material by careful graded reduction.

Assay of sample.....Gold.....2.22 ounces per ton.

Purpose of Shipment.—Information was desired as to the best method of treatment of the ore for the design of a mill to recover the gold.

Scope of Experimental Work.—There were five possible methods of treatment, namely: amalgamation; amalgamation followed by concentration; amalgamation followed by concentration and cyaniding; amalgamation followed by cyaniding; and the simple method of straight cyaniding. In order to obtain necessary data for comparing the merits of each, tests were conducted using each method.

EXPERIMENTAL TESTS

Test No. 1

Amalgamation.—A sample of the ore was crushed to 35 mesh and amalgamated.

Procedure.—1,000 grammes of ore was agitated with 100 grammes of mercury in a pulp of 1: 1 density for 2 hours.

Results—

Assay of heads.....	2.22 ounces per ton
Assay of amalgamation tailing.....	0.52 “
Recovery by amalgamation.....	76.4 per cent

Test No. 2

Amalgamation followed by Table Concentration.—The amalgamation tailing from test No. 1 was tabled:—

Product	Weight grammes	Weight per cent	Assay oz./ton	Recoveries per cent
Table concentrate.....	30.4	3.06	6.76	39.20
Table tailing.....	963.4	96.94	0.33	60.80
Heads.....	993.8		0.52	

Recapitulation—

Recovery by amalgamation.....	76.4 per cent.
Recovery in table concentrate.....	9.25 “
Total recovery.....	85.65 “
Loss in tailing.....	14.35 “

Test No. 3

Amalgamation, Table Concentration, and Cyaniding.—The table tailing from test No. 2 was cyanided, the tailing first being recrushed to pass 100 mesh. A cyanide solution of 0.25 per cent potassium cyanide and lime 10 pounds per ton was used. Time of agitation was 24 hours.

Assay of table tailing before cyaniding.....	0.33 ounce per ton.
Assay of cyanide tailing.....	0.02 “
Extraction by cyaniding.....	94.0 per cent.

Recapitulation—

Recovery by amalgamation.....	76.4 per cent.
Recovery in table concentrate.....	9.25 “
Extraction by cyaniding.....	13.5 “
Total recovery.....	99.15 “

Test No. 4

Amalgamation followed by Cyanidation of Amalgamation Tailing.—A new sample was crushed to 35 mesh and amalgamated as in test No. 1.

Assay of heads.....	2.22 ounces per ton.
Assay of amalgamation tailing.....	0.56 “
Recovery by amalgamation.....	74.7 per cent.

The amalgamation tailing was recrushed to pass 100 mesh and cyanided. The strength of solution used was 0.25 per cent potassium cyanide; 10 pounds lime per ton was used. The time of agitation was 24 hours.

Assay of amalgamation tailing before cyaniding.....	0.56 ounce per ton.
Assay of cyanide tailing.....	0.02 “
Extraction by cyaniding.....	96.5 per cent.

Recapitulation—

Recovery by amalgamation.....	74.7 per cent.
Extraction by cyaniding.....	24.4 “
Total recovery.....	99.1 “

Test No. 5

Cyaniding.—A sample of ore was crushed to 100 mesh and cyanided. A solution strength of 0.25 per cent cyanide was used; 10 pounds lime per ton added; the time of treatment was 24 hours.

Assay of heads before cyaniding.....	2.22 ounces per ton
Assay of cyanide tailing.....	0.02 “
Extraction by cyaniding.....	99.1 per cent.

SUMMARY AND CONCLUSIONS

1. *Amalgamation.*—76.4 per cent of the gold values were recovered by amalgamation. This is probably higher than what could be expected from actual milling operations, but consideration should be given to amalgamation as a preliminary operation for the recovery of a portion of the gold values.

2. *Amalgamation followed by Table Concentration.*—A further recovery of 9.25 per cent of the gold values is made in a table concentrate, making a total recovery of 85.65 per cent by amalgamation and tabling of the tailing. The concentrate would require further treatment for the recovery of the gold, and the tailing is too high to be discarded. Some subsequent treatment of the tailing is necessary to recover more of the gold.

3. *Amalgamation, Table Concentration, and Cyaniding.*—By this method a total recovery of 99.15 per cent of the gold values is obtained. There is produced, however, a table concentrate requiring further treatment of regrinding and cyaniding, or disposal to the smelters.

4. *Amalgamation and Cyaniding.*—By this method a total recovery of 99.1 per cent of the gold values is obtained, showing that it is not necessary to make a table concentrate.

5. *Straight Cyaniding.*—By this method a recovery of 99.1 per cent of the gold values is obtained, showing that on the sample submitted, a recovery is made equally as good as by any of the above methods.

The choice of methods is, therefore, between amalgamation followed by cyaniding the amalgamation tailing, and straight cyaniding. Both methods give the same recoveries of the gold values. The method adopted will depend on many factors, such as local conditions at the mine, financial conditions, and the general characteristics of the ore-bodies, etc. If coarse gold is present in the ore in any appreciable amounts, it would seem advisable to amalgamate preliminary to cyaniding. It provides for quick returns from a portion of gold values in the ore. After operating for some time, it can be eliminated from the flow-sheet if found unnecessary. Straight cyaniding would be the simpler and cheaper process both as regards cost of installation and operation. The cyanide consumption should not be high, as the sample shows the ore to be remarkably clean.

Report No. 210

THE CLEANING OF MICA PREPARATORY TO TRIMMING AND SPLITTING

R. K. Carnochan

Shipment.—Two barrels of mica, net weight 564 pounds, were received January 28, 1924, from Blackburn Bros., Ottawa.

Characteristics of the Mica.—The mica received was amber mica in pieces ranging in size from about 4 by 3 inches by $\frac{1}{2}$ inch thick down to very small flakes. All of the larger pieces had small thin flakes of mica adhering to their faces. A small amount of rock was present in the shipment.

Purpose of Experimental Tests.—Tests were desired to see what could be done to remove the unattached fine flakes and to loosen and remove the adhering fine particles from the faces of the larger flakes. At present it is customary to split off a layer of mica from each face of the larger pieces so as to remove all adhering matter. It was thought that this method was wasteful, and might be improved on.

Arrangements for Experimental Tests.—Mr. H. L. Forbes of Blackburn Bros. Ltd., was present during most of the test work and gave valuable advice.

Test No. 1 (Suction)

A lot of 167 $\frac{1}{2}$ pounds of mica was dried and then run over a 1-inch mesh Ferraris shaking screen. A suction pipe leading to a large collector was arranged at the discharge end of the screen to remove any thin flakes of mica that were too large to pass through the 1-inch mesh screen. A fan was used to create the suction. A small amount of rock was picked off the screen by hand.

Product	Weight pounds	Weight per cent
Rock.....	6.5	3.9
Oversize, +1 inch.....	134.0	80.0
Suction +1 inch.....	3.5	2.1
Fines, -1 inch.....	18.0	10.7
Loss.....	3.0	1.8
Moisture.....	2.5	1.5
Heads.....	167.5	100.0

Test No. 2 (Washing and Suction)

A lot of 283 pounds of mica was washed in a small revolving trommel fitted with a $\frac{1}{2}$ -inch screen. The mica was placed in the trommel, which was closed at both ends, through a small door. The door was closed and the trommel was revolved while half-submerged in a tank of water. The faces of the larger pieces were cleaned and the finer flakes passed through the screen and settled to the bottom of the tank. After washing, the mica was dried and run over the Ferraris screen fitted with a suction pipe as described under test No. 1.

Product	Weight, pounds	Weight, per cent
Rock.....	12.5	4.4
Washings, - $\frac{1}{2}$ inch.....	45.0	15.9
Oversize, +1 inch.....	152.5	53.9
Suction, +1 inch.....	12.5	4.4
Fines, -1 inch.....	46.5	16.5
Loss.....	10.0	3.5
Moisture in heads.....	4.0	1.4
Heads.....	283.0	100.0

Test No. 3 (Suction, Washing and Suction)

A lot of 113.5 pounds of mica was dried and put over the 1-inch Ferraris shaking screen fitted with a suction pipe. The oversize +1-inch product was washed in the revolving trommel. The washed product after

drying, was run over the 1-inch Ferraris shaking screen fitted with a suction pipe. Similar products from the two runs on the Ferraris screen were combined.

Product	Weight, pounds	Weight, per cent
Rock.....	3.5	3.1
Washings, $-\frac{1}{2}$ inch.....	15.0	13.2
Oversize, +1 inch.....	63.5	55.9
Suction, +1 inch.....	5.0	4.4
Fines, -1 inch.....	22.0	19.4
Loss.....	3.0	2.7
Moisture in heads.....	1.5	1.3
Heads.....	113.5	100.0

Summary of Experimental Tests

Product	Test No. 1 Weight per cent	Test No. 2 Weight per cent	Test No. 3 Weight per cent
Rock.....	3.9	4.4	3.1
Washings, $-\frac{1}{2}$ inch.....		15.9	13.2
Oversize, +1 inch.....	80.0	53.9	55.9
Suction, +1 inch.....	2.1	4.4	4.4
Fines, -1 inch.....	10.7	16.5	19.4
Loss.....	1.8	3.5	2.7
Moisture.....	1.5	1.4	1.3

CONCLUSIONS

Running the mica over the 1-inch Ferraris shaking screen fitted with a suction pipe is very beneficial. All the free fine mica is removed and even some of that adhering to the faces of the larger pieces. This results in an oversize +1-inch product that would be ideal for trimming, except that there is still a small amount of fine mica adhering to the faces of the larger pieces. The fines from the screen would be suitable for making ground mica. The suction product consists of flake mica all +1 inch and ranging up to about 2 by 2 inches. Some of it is rather too thick to allow the whole product to be sold as split mica, but it seems quite probable that either by more careful adjustment of the size of screen and amount of suction used, or re-running, a suction product could be made that would sell as split mica.

Washing in the trommel followed by treatment in the shaking screen with suction results in a very nice oversize product quite free of any small pieces of mica adhering to the faces of the larger flakes, it has the disadvantage, though, of cutting down the percentage of oversize from 80 to 55. The trommel crushes the mica up as well as washes it, and on this account cannot be recommended. If, instead of washing in the trommel the washing were done by passing the mica under water sprays on a moving belt and some suitable arrangement were used to turn the mica over, so as to wash both sides, it is quite likely that a large percentage of oversize free from adhering particles would be obtained.

Report No. 211

MINERALIZED DIKE MATERIAL FROM ASSOCIATED GOLDFIELDS, LTD.,
LARDER LAKE, ONTARIO

J. S. Godard

Shipment.—Two small shipments were received during April, 1924, from Dr. H. C. Cooke of the Geological Survey, Ottawa. Shipment No. 1 consisted of 4 pounds, and shipment No. 2 of 7 pounds.

Purpose of Experimental Tests.—The object of the test work was to determine whether the gold values were associated with the arsenopyrite or the pyrite.

Characteristics of the Shipment.—The shipments submitted were designated "dike ore" and were supposed to be typical of a large body containing arsenopyrite and pyrite as the chief sulphide minerals.

Analyses :

	Shipment No. 1	Shipment No. 2
Gold.....	0.12 ounce per ton	0.10 ounce per ton
Arsenic.....	1.33 per cent	0.86 per cent
Iron.....	9.90 per cent	7.98 per cent

EXPERIMENTAL TESTS

A number of tests were made to obtain a separation of the arsenopyrite from the pyrite. These consisted of flotation tests; sizing and tabling of the sized material, hydraulic classification, air jigging, electrostatic separation, and roasting of the material at a temperature below the decomposition point of pyrite. None of these were successful with the exception of tabling of closely-sized material. A separation was noticed on the table, but there was not sufficient material in shipment No. 1 to obtain clean products for determination of the gold content. A further shipment was requested and shipment No. 2 consisting of 7 pounds was received.

Shipment No. 2 was crushed to 35 mesh and sized on 100- and 200-mesh screens. The sized products were tabled making a sulphide concentrate and a gangue tailing. The ratio of concentration was 1 : 13.5 showing the rock to contain about 7.7 per cent sulphides. The sulphide concentrates were closely sized on 48-, 65-, 80-, 100-, 115-, 150-, 170- and 200-mesh Tyler standard screens. Each size was tabled one after the other and the middling products re-run. By this close sizing and tabling four products were obtained, grading from practically pure arsenopyrite in No. 1 to pyrite in No. 4.

Results of Experimental Tests.—Each product was assayed with the following results:—

Product	Weight grms.	Arsenic per cent	Gold oz/ton
No. 1 concentrate.....	2.05	24.04	1.29
No. 2 "	8.50	22.70	0.80
No. 3 "	11.00	15.71	0.96
No. 4 "	157.20	7.19	0.70

CONCLUSIONS

The above results show that the gold values are associated with both the arsenopyrite and the pyrite, but to a greater extent with the arsenopyrite. The table tailings contained 0.03 ounce per ton gold.

Report No. 212

THE CONCENTRATION OF GOLD-COPPER ORE FROM SPROAT LAKE, NEAR ALBERNI, BRITISH COLUMBIA

C. S. Parsons

Shipment.—A shipment of 25 pounds of gold-copper ore was received January 18, 1924, from Messrs. Smith and Beavan, Alberni, B.C. The sample was from their property at Sproat lake, near Alberni.

Characteristics of the Ore.—The sample submitted was a sulphide ore consisting of chalcopyrite associated with iron pyrite in a siliceous gangue. The copper and iron sulphides carry the gold and silver values.

Purpose of Experimental Tests.—The object in conducting experimental tests was to determine whether the ore could be concentrated with the production of a marketable concentrate containing the copper and precious metal values.

Sampling and Analysis.—The shipment was sampled by stage-crushing and cutting in a Jones riffle sampler. Analysis of the sample showed it to contain:—

Copper.....	1.22 per cent	Gold.....	0.34 ounce per ton
Arsenic.....	trace	Silver.....	1.06 ounce per ton
Sulphur.....	9.57 per cent	Iron.....	8.69 per cent
Insoluble.....	76.52 per cent		

EXPERIMENTAL TESTS

Test No. 1—Selective Flotation

Object of Test.—The purpose of this test was to determine whether the gold was with the copper mineral only, and if such were the case, to produce a high-grade copper concentrate carrying the gold, by the elimination of the iron sulphide with gangue.

Procedure.—1,000 grammes of the ore was crushed in a ball mill to pass 65 mesh and floated in a laboratory Ruth flotation machine. The following reagents were fed to the ball mill with the ore:—

Lime.....	5.0 pounds per ton
Coal tar and coal-tar creosote mixture.....	0.5 pounds per ton

Lime was added to prevent the iron sulphide from floating with the copper.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Cu Per cent	Au oz/ton	Cu	Au
Concentrate.....	9.0	11.36	1.22	91.9	37.6
Tailing.....	91.0	0.10	0.20	8.1	62.4

Conclusions.—First: the results given in the above table show that only part of the gold is associated with the copper. Second: that the use of lime will prevent the iron sulphides from floating and will give a high-grade copper concentrate.

Test No. 2—Straight Flotation

Object of Test.—The purpose of this test was to float all the sulphides in the ore.

Procedure.—1,000 grammes of the ore was crushed in a ball mill to pass 65 mesh and floated in a laboratory Ruth flotation machine. A coal tar and coal-tar creosote mixture, 0.5 pound per ton of ore, was added at the ball mill.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Cu per cent	Au oz/ton	Cu	Au
Concentrate.....	23.6	4.76	1.12	98.7	89.7
Tailing.....	76.4	0.02	0.04	1.3	10.3

Conclusions.—First: the results of this test show that a high recovery of the gold can be obtained by the flotation of the sulphides. Second: that the gold is associated with both the copper and iron sulphides. Third: that the sulphide content of the ore can be concentrated by straight flotation in a neutral pulp.

Test No. 3.—Straight Flotation Followed by Tabling

Object of Test.—This test was made to determine whether the addition of acid to the flotation pulp would float more sulphide and increase the recovery of the gold, as the results of the previous tests indicated that the gold was associated with the sulphides.

Procedure.—1,000 grammes of the ore was crushed to 65 mesh in a ball mill and the following reagents used:

Coal tar and coal-tar creosote, 0.5 pound per ton added to ball mill
Sulphuric acid, 10.0 pounds per ton added to flotation cell.

Results.—The results of the test are given in the following table:

Product	Weight per cent	Analysis		Per cent of values	
		Cu per cent	Au oz/ton	Cu	Au
Flotation concentrate.....	25.2	4.86	1.24	98.2	79.3
Table concentrate.....	0.5	8.86	11.2
Table tailing.....	74.3	0.03	0.05	1.8	9.5

Combined concentrates—Concentration ratio 3.89 : 1

Analysis—Copper, 4.86 per cent; gold, 1.396 ounces per ton.

Recoveries—Copper, 98.2 per cent; gold, 90.5 per cent.

Conclusions.—First: a high recovery of the gold and copper is obtained by flotation and tabling but the grade of concentrate is low. Second: the use of acid increases the recovery of the sulphides by flotation but the flotation tailing is higher in gold than in test No. 2, where the flotation

was made in a neutral pulp without the addition of lime or acid. Third: the high-grade gold concentrate obtained from the tabling indicates the presence of some free gold.

Test No. 4.—Selective Flotation followed by Tabling

Object of Test.—The test was made to determine whether a concentrate containing a higher content of copper could be obtained by first producing a high-grade copper concentrate by selective flotation and then tabling the flotation tailing for the recovery of the gold not reporting with the copper flotation concentrate.

Procedure.—1,000 grammes of the ore was crushed to 65 mesh in a ball mill. The following reagents were added in the ball mill:—

Lime.....	5.0 pounds per ton
Coal tar and coal-tar creosote.....	0.5 pound per ton
Steam-distilled pine oil.....	0.05 pound per ton

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Cu per cent	Au oz/ton	Cu	Au
Flotation concentrate.....	8.9	12.66	1.12	92.6	30.0
Table concentrate.....	13.8	0.20	1.24	2.3	51.4
Table tailing.....	77.3	0.08	0.08	5.1	18.6

Combined concentrates—Concentration ratio 4.4 : 1.

Analysis—Copper, 5.08 per cent; gold, 1.19 ounces per ton.

Recoveries—Copper, 94.9 per cent; gold, 81.4 per cent.

Conclusions.—First: a high-grade copper concentrate can be made by selective flotation in an alkaline pulp, but the recovery of the gold is very low. Second: the tabling of the tailing from the selective flotation test did not give so high a total recovery as was obtained in the preceding test No. 3. The reason for this was due to the table not saving the fine iron sulphides which carry the gold and which were thrown into the flotation tailing by the selective action of the alkaline pulp.

Test No. 5—Amalgamation

Object of Test.—An amalgamation test was run to determine whether there was any free gold present in the ore.

Results—

Heads before amalgamation.....	0.34 ounce per ton
Tailing after amalgamation.....	0.32 ounce per ton
Recovery by amalgamation.....	0.02 ounce per ton = 5.9 per cent

Conclusions.—The above results show that there is very little free gold present in the ore and that the gold is associated with and contained in the copper and iron sulphides.

Summary.—A concentrate containing 4.86 per cent copper and 1.4 ounces per ton gold can be obtained with a concentration ratio of 3.9 : 1, and a recovery of 98.2 per cent of the copper and 90.5 per cent of the gold

values. This was obtained by flotation followed by tabling the flotation tailing as in test No. 3.

A comparison of the flotation results from the four tests shows that selective flotation combined with tabling does not give so high a recovery of the gold values as straight flotation in a neutral or acid pulp. Flotation in a neutral pulp, as in test No. 2, gives a higher recovery of the gold values by flotation alone than flotation in an acid pulp, as in test No. 3. The amalgamation test shows that very little gold, not over 5.9 per cent of the total amount present, could be recovered by introducing amalgamation into the flow-sheet.

CONCLUSIONS AND RECOMMENDATIONS

The tests show conclusively that the gold is associated with the sulphides, and very little, if any, is free milling. This means that there will be an equivalent loss of gold for any loss of sulphide in the tailing, or in other words, a high recovery of the sulphides means a high recovery of the gold.

The procedure followed in the above tests was to grind the ore to the required fineness for flotation and to table the flotation tailing. As the results of these tests show that to obtain a high recovery of the gold values a high recovery of all the sulphides in the ore is necessary, it therefore would make no material difference to the recoveries of the copper and gold values whether tabling was done before or after flotation.

Tabling before flotation will require a more extensive plant, and the cost of operation will be higher. Classification of the pulp for table feed, regrinding of the coarse table tailing, and thickening of the slimes for flotation would be necessary, requiring more floor space with increased cost of installation, operation, and maintenance. The efficiency of flotation would not be so high as on freshly crushed ore feed.

By using flotation followed by tabling, the mill will be compact, and require considerably less floor space. The grinding can be done in one operation in ball mills in closed circuit with drag classifiers, the ball mill receiving a 1½-inch feed, and the discharge from the circuit going direct to the flotation cells. The flotation tailings will pass straight to the tables which will only have to handle a small amount of coarse sulphide that the flotation cells fail to pick up. These tables can be crowded and their capacity will be double the capacity of tables working on the crude ore feed containing all the sulphides. The number of tables required will be about half that necessary for tabling before flotation. The flotation operator would be able to look after the tables as well as the flotation cells. If Callow flat-bottom flotation cells are used, the cleaning and roughing can be done in the one cell, thus saving considerable head-room in the mill.

Report No. 213

METALLURGICAL TESTS ON GOLD ORE FROM THE REX MINE, HERB LAKE,
NORTHERN MANITOBA

C. S. Parsons

Shipment.—A shipment of 142 pounds of gold ore consisting of four separate samples was received April 4, 1924, from the Manitoba Metals Co., Herb lake, Man. The samples were supposed to be representative

of the ore of the Rex mine, which is being developed by the above company, a subsidiary of the Mining Corporation of Canada.

Characteristics of the Ore.—The samples submitted consisted of white quartz containing free gold visible to the naked eye, and small amounts of sulphides, chiefly iron pyrite.

Purpose of Experimental Tests.—The object in conducting the experimental tests was to determine a simple process for the recovery in a small plant of the values in the ore broken during development, and to determine the amenability of the ore to various methods of treatment.

Analyses of Samples.—The weights and assays of the samples received were as follows:—

Sample mark	Weight lbs.	Gold oz./ton	Silver oz./ton
Main sample.....	89.0	1.096	0.17
Bag No. 312.....	11.0	0.90	0.17
“ “ 362.....	21.5	0.945	0.17
“ “ 370.....	20.0	0.90	0.17

NOTE.—No particular difference was noticed in the characteristics of the ore of all four samples, and the assays show that they were all of practically the same grade.

EXPERIMENTAL TESTS

Test No. 1.—Amalgamation

An amalgamation test was made on 38 pounds of ore taken from the main sample.

Description of Test.—The ore was crushed dry to 28-mesh Tyler standard screen and fed through a mercury-trap amalgamator on to amalgamation plates at the rate of one ton of ore to 2.5 square feet of plate area.

Results—

Heads before amalgamation.....	1.096 ounces per ton
Tailing after amalgamation.....	0.16 ounce per ton
Recovery of gold values.....	85.4 per cent

Test No. 2—Blanket Test

A blanket test was made on 49 pounds of ore from the main head sample.

Description of Test.—The ore was crushed dry to 28-mesh Tyler standard screen, and fed over corduroy blankets at the rate of 1.1 tons of ore to 2.5 square feet of blanket area.

Results—

Heads to blankets.....	1.096 ounces per ton
Tailing from blankets.....	0.52 ounce per ton
Blanket concentrate obtained.....	882.4 grammes
Assay of blanket concentrate.....	15.45 ounces per ton
Gold values in blanket concentrate.....	56.2 per cent
Gold values remaining in tailing.....	43.8 per cent

Test No. 3.—Table Test on Amalgamation Tailing

A 5,000-gramme sample of the amalgamation tailing from test No. 1, was tailed on a laboratory Wilfley table. The results were as follows:—

Product	Weight		Assay Au oz/ton	Per cent of gold values
	Grammes	Per cent		
Table concentrate.....	68.7	1.4	7.35	49.0
Table tailing.....	4768.0	98.6	0.11	51.0

A very clean table concentrate was made. It is quite probable that a much lower tailing could be obtained by cutting out more concentrate. The total recovery by amalgamation and tabling was approximately 93 per cent.

Test No. 7.—Amalgamation Test on Blanket Concentrate

Description of Test.—685 grammes of blanket concentrate from test No. 2 was amalgamated with 10 per cent by weight of mercury in a pulp density of 50 per cent solids.

Results—

Assay before amalgamation.....	15.45 ounces per ton
Assay after amalgamation.....	1.20 ounces per ton
Recovery of gold values.....	92.3 per cent.

Test No. 8.—Amalgamation Test on Table Concentrate

Description of Test.—40 grammes of table concentrate from test No. 3 was amalgamated in a mortar. The mercury was panned out and the tailing assayed.

Results—

Assay before amalgamation.....	7.35 ounces per ton
Assay after amalgamation.....	1.03 ounces per ton
Recovery of gold values.....	86.0 per cent.

Cyanide Tests on Amalgamation Tailing.—Tests Nos. 9 and 10 were run on amalgamation tailing containing 0.24 ounce gold per ton. The details of the tests and the results obtained are given below:—

Test No. 9—Part 1

	A	B	C
Degree of grinding, mesh.....	—65	—100	—150
Weight of tailing, grammes.....	200	200	200
Water, c.c.....	400	400	400
Potassium cyanide, per cent solution.....	0.25	0.25	0.25
Lime, pounds per ton.....	5	5	5
Time of agitation, hours.....	24	24	24
Results—			
Cyanide heads, ounce per ton.....	0.24	0.24	0.24
Cyanide tailing, ounce per ton.....	0.008	0.005	0.005
Recovery, per cent.....	96.8	98.0	98.0
Cyanide consumption, pounds per ton.....	0.866	0.91	0.56
Lime consumption, pounds per ton.....	3.9	4.2	3.72

Test No. 9—Part 2

	A	B	C
Degree of grinding, mesh.....	-65	-100	-150
Weight of tailing, grammes.....	200	200	200
Water, c.c.....	400	400	400
Potassium cyanide, per cent solution.....	0.25	0.25	0.25
Lime, pounds per ton.....	5	5	5
Time, hours.....	48	48	48
Results—			
Cyanide heads, ounce per ton.....	0.24	0.24	0.24
Cyanide tailing, ounce per ton.....	0.01	0.01	0.01
Recovery, per cent.....	95.8	95.8	95.8
Cyanide consumption, pounds per ton.....	0.62	0.98	0.786
Lime consumption, pounds per ton.....	3.8	4.18	4.02

Test No. 10—Part 1

	A	B	C
Degree of grinding, mesh.....	-65	-100	-150
Weight of tailing, grammes.....	200	200	200
Water, c.c.....	600	600	600
Potassium cyanide, per cent solution.....	0.05	0.05	0.05
Lime, pounds per ton.....	5	5	5
Time, hours.....	24	24	24
Results—			
Cyanide heads, ounce per ton.....	0.24	0.24	0.24
Cyanide tailing, ounce per ton.....	0.05	0.02	0.02
Recovery, per cent.....	79.2	91.7	91.7
Cyanide consumption, pounds per ton.....	0.632	1.23	1.11
Lime consumption, pounds per ton.....	3.54	3.81	3.56

Test No. 10—Part 2

	A	B	C
Degree of grinding, mesh.....	-65	-100	-150
Weight of tailing, grammes.....	200	200	200
Water, c.c.....	400	400	400
Potassium cyanide, per cent solution.....	0.05	0.05	0.05
Lime, pounds per ton.....	5	5	5
Time, hours.....	48	48	48
Results—			
Cyanide heads, ounce per ton.....	0.24	0.24	0.24
Cyanide tailing, ounce per ton.....	0.04	0.01	0.01
Recovery, per cent.....	84.4	95.8	95.8
Cyanide consumption, pounds per ton.....		0.4	0.91
Lime consumption, pounds per ton.....		3.67	4.19

SUMMARY AND CONCLUSIONS

The tests show a recovery of the gold values of 85 per cent by amalgamation alone. Amalgamation shows a much higher recovery than concentration on corduroy blankets and amalgamation of the blanket concentrate.

Tabling after amalgamation and amalgamation of the table concentrate increases the recovery of the gold values.

Grinding to 30 mesh for amalgamation gives a high recovery of the gold values. The condition of the mercury was good, both inside and on the plates, showing that the ore is very amenable to amalgamation.

Cyanide tests on the amalgamation tailing show very high recoveries of the remaining gold values after amalgamation.

Report No. 214

EXAMINATION OF AND EXPERIMENTAL TESTS ON GOLD ORE FROM
DOMINION CLAIMS, COPPER LAKE, NORTHWESTERN MANITOBA

W. B. Timm

Shipments.—Two small shipments of gold ore were received from A. L. Stewart, Esq., 280 Smith St., Winnipeg, Man. Shipment No. 1 consisting of two small pieces of ore weighing approximately $\frac{1}{2}$ pound was received on February 27, 1924. Shipment No. 2 consisting of two samples of the ore, weighing $6\frac{1}{2}$ pounds and $\frac{1}{2}$ pound respectively, was received April 29, 1924.

Location of Claims.—The samples submitted were taken from a 14-inch stringer in an ore zone of 32 feet in width on the Dominion claims on Copper lake. The claims are situated approximately 80 miles north of The Pas, Man., and 12 miles northeast of the proposed railway to the Flin-Flon camp.

Characteristics of the Ore.—The samples consisted of mineralized quartz, the chief minerals being iron pyrites and small amounts of galena. No free gold was visible in the samples of No. 1 shipment, but in the samples of No. 2 shipment free gold which showed up as metallics in grinding the samples for analysis and experimental tests was detected.

Purpose of Examination and Experimental Tests.—An examination of the samples was requested to determine in what form the gold and silver values occurred in the ore. Panning tests made on the ore by the owners revealed no gold, although similar samples gave high values in gold on assay.

Analysis of Ore Samples.—The samples received gave the following values on assay:—

	Gold oz/ton	Silver oz/ton	Tellurium %
Shipment No. 1.....	13.92	45.56	0.09
Shipment No. 2, large sample.....	4.91	16.37	
Shipment No. 2, small sample.....	7.50	23.47	

Experimental Tests on Shipment No. 1

The two small pieces of ore were crushed to 40 mesh and screened on 80 mesh. This gave: —40+80 83 grammes
—80 126 “

Each of the above sizes was tabled separately on a laboratory Wilfley table and the combined table tailings were floated in a laboratory Ruth flotation machine. The results of this test were as follows:—

Product	Weight grms.	Gold oz/ton	Silver oz/ton	Tellurium per cent
Ore.....		13.92	45.56	0.09
+80 table concentrate.....	10.3	9.80	27.00	
—80 table concentrate.....	6.6	84.40	150.80	
Flotation concentrate.....	4.7	140.00	380.30	
Flotation middling.....	47.0	10.32	45.48	
Flotation tailing.....	144.2	3.28	16.40	

The table concentrates were practically clean sulphides, mostly pyrite, whereas the flotation concentrate contained very little mineral. The flotation middling and tailing consisted of quartz gangue, very little mineral being noticeable on examination under the microscope.

Conclusions from Results of the above Test.—No free gold was visible and no metallics were obtained in preparing the sample for analysis and for the above test, which indicated that if the gold were present as free gold it was in a very finely divided state. The presence of tellurium in the ore as indicated by the analysis suggests that the gold may occur as the telluride.

The coarse table concentrate, +80 mesh, which is practically clean sulphides, gave a lower assay than the ore, and the lowest of any of the concentration products with the exception of the tailings. This would indicate that the iron pyrite is not carrying the gold values. If it were, there would be a concentration of the gold in this product.

The -80 mesh table concentrate and the flotation concentrate, which would be 200 mesh and finer, show the highest values in both gold and silver. This would indicate that in whatever form the values occur, they are in a finely divided state.

Microscopic Examination of Products from Shipment No. 1.—The -80-mesh table concentrate and the flotation concentrate carrying the high gold values were examined under a high power microscope. No free gold could be detected. There was not sufficient of the products available to make a thorough examination. It is, however, very probable that the gold is present as tellurides. Larger samples are required for a more extended examination in order to reach a definite conclusion on this point.

Experimental Tests on Shipment No. 2

A concentration test by tabling and flotation, an amalgamation test and two cyanide tests were made on the larger sample of ore submitted with shipment No. 2.

Concentration Test.—This test was conducted to determine in what products the gold and silver values were reporting and to obtain products for microscopic and chemical examination. 1,000 grammes of the ore was crushed to 80 mesh. In crushing the sample for assay some metallics were obtained and free gold was observed in the ore. The -80 mesh material was sized on 100-, 150-, and 200-mesh Tyler standard screens. Each size was tabled on a laboratory Wilfley table and the combined tailings floated in a laboratory Ruth flotation machine. The results of this test were as follows:—

Product	Weight grms.	Gold oz./ton	Silver oz./ton
Table concentrate +100.....	13.1	10.21	22.45
Table concentrate +150.....	8.9	14.29	20.90
Table concentrate +200.....	7.2	28.58	60.66
Table concentrate -200.....	9.0	144.90	226.00
Flotation concentrate.....	11.8	105.80	322.00
Flotation middling.....	83.0	6.76	34.74
Flotation tailing.....	784.0	0.60	2.76
Loss in test.....	83.0
Ore.....	1000.0	4.91	16.37

Amalgamation Test.—This test was conducted to determine the percentage of gold present as free gold that would amalgamate; 40 per cent of the gold values and 7 per cent of the silver values were recovered from the mercury used in the test. This checked with the assay of the amalgamation tailing. The test was made on the ore ground to 200 mesh, in order to obtain the maximum recovery by amalgamation.

Cyanide Tests.—Two cyanide tests were made on the ore ground to 200 mesh. One was given 48 hours' agitation, the other 96 hours. The results were as follows:—

Tailing assay test	No. 1—48 hours' agitation.....	Gold, 0.08;	silver, 0.78
	" 2—96 " " " " " " " " " " " "	" 0.06;	" 0.80
Recoveries by cyanidation.....	" " " " " " " " " " " "	" 98%;	" 95%

Conclusions from Results of above Tests on Shipment No. 2.—The ore of shipment No. 2 contains free gold. Small particles were detected in the ore visible to the naked eye, and more so under the microscope. Metallic gold was obtained in preparing the sample for analysis. The amalgamation test showed that 40 per cent of the gold values and 7 per cent of the silver values were recovered by amalgamation. This test also showed that the silver was not alloyed with the gold to any extent, and that it was not present as native silver, but that it was either associated with the galena in the ore, or as a separate mineral, possibly argentite or ruby silver. The low recovery of the gold values by amalgamation could only be accounted for by the remaining gold being either locked up with the sulphides in a very finely divided state or present in the form of tellurides.

The results of the concentration test on shipment No. 2 were very similar to those on shipment No. 1. The high values in both gold and silver reported in the fine table concentrate and flotation concentrate, showing that the coarse pyrite carried relatively small values as compared with the finer products, and that the gold and silver minerals were present in a very finely divided state.

The high recoveries of both gold and silver obtained in the cyanide tests, and the low consumption of cyanide and lime, indicated that the ore contained no objectionable impurities; that the gold and silver minerals were in a finely divided state, and readily soluble in cyanide solution. The small amounts of tellurium, arsenic, and antimony present had but little effect on the solubility of the gold and silver.

Chemical Tests.—To determine the presence of tellurium in the ore, a qualitative analysis was made on 50 grammes. Tellurium was determined in small amounts, indicating that a portion of the gold values was present in the form of gold tellurides.

To determine the presence of ruby silver in the ore, a qualitative analysis was made for arsenic and antimony in the ore, in the flotation concentrate which contained the high silver values, and in the +200 mesh table concentrate. Very small amounts of arsenic were present in all three, but much more antimony was present in the flotation concentrates than in the ore or in the +200 table concentrate. The +200 table concentrate contained a much larger proportion of iron pyrite than the flotation concentrate. The higher antimony content in the flotation concentrate would indicate that the antimony was not associated with the pyrite but with the silver mineral in the flotation concentrate, indicating the presence of ruby silver.

Microscopic Examination.—Free gold was detected in the ore and in the concentration products. Ruby silver was detected in the ore and in the flotation concentrate. Although no tellurides could be detected in any of the products, the chemical analysis shows the presence of tellurium beyond a doubt.

SUMMARY OF EXAMINATION AND EXPERIMENTAL TESTS

The ore submitted for examination and test contains free gold. It also contains tellurium, indicating the presence of gold tellurides. The silver is present as ruby silver, a small portion may be present associated with the small amount of galena in the ore, and a portion may be present as silver sulphide, argentite, although this mineral was not detected by microscopic examination.

The ore is amenable to treatment by cyanidation, with high recoveries of the gold and silver values and low consumption of cyanide and lime.

Report No. 214a

EXPERIMENTAL TESTS ON GOLD ORE FROM DOMINION CLAIMS ON COPPER LAKE, NORTHWESTERN MANITOBA

J. S. Godard

Shipment.—A further shipment of 198 pounds of ore was received September 3, 1924, from A. L. Stewart, Esq., 280 Smith street, Winnipeg.

Purpose of Shipment.—The shipment was made to check the accuracy of the results obtained from experimental tests on former small shipments received for examination as to the form in which the gold and silver values occurred in the ore.

Sampling and Analysis.—The ore was crushed to half inch and cut once in a riffle sampler. One half was ground to 14-mesh and cut twice in the sampler. One of these portions was ground to pass 48 mesh. The sample for analysis was cut from this portion and gave the following:—

Gold.....	2.88 ounces per ton
Silver.....	11.14 ounces per ton

EXPERIMENTAL TESTS

A series of amalgamation tests was made to determine the amount of gold and silver values that could be recovered by amalgamation at various stages of grinding. For each test 1,000 grammes of ore was used with 100 grammes of mercury, and a pulp density of 50 per cent solids. The contact period in each case was two hours. The amalgamation tailings were screened and the products assayed. The results are given in the following table.

A series of cyanide tests was made to determine the amenability of the ore to treatment by the cyanide process. The accompanying table shows the details of each test and the results obtained.

Amalgamation Tests

Test No.	Screen analysis of tailings				Per cent of values		Per cent amalgamated	
	Mesh	Weight per cent	Assay oz. per ton					
			Gold	Silver	Gold	Silver	Gold	Silver
2	+100.....	3.9	1.26	5.55	2.5	2.1	32.6	0.64
	—100+150.....	13.4	1.24	5.39	8.5	6.9
	—150+200.....	17.2	2.61	6.06	23.2	10.0
	—200.....	65.5	1.95	12.92	65.8	81.0
4	+100.....	1.0	2.57	9.20	1.7	0.9
	—100+150.....	6.3	1.39	5.77	5.6	3.7	45.4	11.4
	—150+200.....	12.6	1.11	5.19	8.8	6.6
	—200.....	80.1	1.65	10.95	83.9	88.8
6	+150.....	2.3	1.81	6.81	3.0	1.7
	—150+200.....	11.5	1.11	4.71	9.0	5.8	51.4	15.6
	—200.....	86.2	1.43	10.10	88.0	92.5
23	+48.....	6.0	2.37	7.1
	—48+65.....	7.1	2.33	8.2
	—65+100.....	15.0	1.88	14.1	30.2
	—100+150.....	12.4	1.56	9.6
	—150+200.....	11.3	1.32	7.4
	—200.....	48.2	2.23	53.6

SUMMARY

1. *Amalgamation.*—At 48 mesh 30.2 per cent of the gold was recovered by amalgamation. This recovery was increased to 51.4 per cent with ore ground to 150 mesh. Less than 1 per cent of the silver was amalgamated at 100 mesh. This amount increased with fine grinding until a maximum of 15.6 per cent was obtained on 150-mesh material.

2. *Cyanidation.*—The recovery by cyanidation at 48 mesh was 84.1 per cent of the gold and 61.9 per cent of the silver. On material ground to 150 mesh and cyanided with 0.30 per cent potassium cyanide for twenty-four hours, the recovery was 96.5 per cent of the gold and 92.4 per cent of the silver. A small increase in recovery is obtained by increasing the time. Both the cyanide and lime consumption was moderate. Better results were obtained using a dilute pulp.

CONCLUSIONS

The results of the experimental tests on the larger shipment show that the ore is amenable to treatment by cyanidation with high recoveries of the gold and silver values, and that the results check closely with those reported on the former smaller shipments.

Cyanidation Tests

Test No.	Weight grms.	cs l Mesh	Water grms.	Per cent cyanide	Lime lb/ton	Time hours	Heads		Tailing		Per cent recovery		Reagents consumed	
							Gold oz./ton	Silver oz./ton	Gold oz./ton	Silver oz./ton	Gold	Silver	Cyanide lbs/ton	Lime lbs/ton
7	295	-48	590	0.20	5	24	2.88	11.14	0.46	4.24	84.1	61.9	2.3	4.2
8	297	-150	594	0.10	5	24	2.88	11.14	0.36	3.57	87.6	68.0	1.9	4.5
9	299	-200	600	0.20	5	48	2.88	11.14	0.23	0.79	92.1	93.0	3.3	4.5
10	300	-150	600	0.20	5	24	2.88	11.14	0.15	1.26	94.8	88.6	2.5	4.4
10b	300	-150	600	0.20	5	48	2.88	11.14	0.16	0.70	94.4	93.8	2.9	4.3
17	300	-150	900	0.10	4	24	2.88	11.14	0.38	2.24	86.8	80.0	2.1	2.0
18	305	-150	915	0.20	4	24	2.88	11.14	0.16	1.04	94.4	90.6	2.9	2.0
19	303	-150	909	0.30	4	24	2.88	11.14	0.10	0.80	96.6	92.8	3.0	1.9
20	300	-150	900	0.30	4	24	2.88	11.14	0.10	0.84	96.5	92.4	3.0	2.3
21	300	-150	900	0.30	4	48	2.88	11.14	0.07	0.69	97.6	93.8	3.1	2.8
22	300	-150	900	0.30	4	72	2.88	11.14	0.06	0.68	97.9	93.8	3.2	2.5
24	300	-150	300	0.20	5	24	2.88	11.14	0.88	3.60	69.4	67.6	2.4	3.9
25	300	-150	600	0.20	5	24	2.88	11.14	0.36	1.88	87.5	83.1	2.7	3.1
26	300	-150	900	0.20	5	24	2.88	11.14	0.21	1.31	92.7	88.2	2.7	2.3

Report No. 215

CONCENTRATION OF MOLYBDENITE ORE FROM THE MOSS MINE, QUYON, QUE.

C. S. Parsons

Shipments.—Two shipments of molybdenite ore, each a carload lot, weighing 25.3 and 42.3 tons, respectively, were received May 16, and June 11, 1924, from the Moss mine, the property of the Canadian-Wood Molybdenite Co., Quyon, Que.

Purpose of Experimental Tests.—The company were re-opening the mine, and starting milling operations. They had secured a contract from a chemical firm for high-grade molybdenite concentrates and desired a large-scale test run to determine the best practice to follow in the concentration of their ore to produce the highest grade of concentrate with good recoveries.

Characteristics of the Ore.—The Moss mine ore consists of segregations of pyrite, pyrrhotite, fluorite, quartz, and orthoclase in quartz syenite. The iron sulphide content is low in comparison to the silica content, and this feature makes this ore more amenable to concentration than those of high iron content. The molybdenite flake is not large, rarely exceeding one-quarter inch in diameter, and is fairly well disseminated throughout the gangue rock. The rock is quite hard to crush, but grinding need not be carried to extreme fineness as the molybdenite is freed at 35 to 48 mesh, the desirable point for flotation. The cost of preparation of the ore for concentration is, therefore, not excessive.

CONCENTRATION RESULTS

Shipment No. 1

One car of ore, net weight.....	50,600 lb.
Analysis of sample cut from dry ore by Vezin sampler.....	2.21% MoS ₂
Average analysis of daily sample of wet feed to cells.....	2.45% MoS ₂
Average of dry and wet samples.....	2.33% MoS ₂
Content of molybdenite, using average analysis.....	1,178.98 lb.
Amount of concentrate obtained.....	1,213.0 lb.
Analysis of concentrate.....	92.93% MoS ₂
Content of molybdenite in 1,213 pounds of concentrate.....	1,127.24 lb.
Calculated analysis of tailing by difference.....	0.10% MoS ₂
Recovery of molybdenite from average assay and calcul. tailing..	95.6%

Shipment No. 2

One car of ore, net weight.....	84,600 lb.
Analysis of sample cut from dry ore by Vezin sampler.....	2.33% MoS ₂
Average analysis, daily sample of wet feed to cells.....	2.60% MoS ₂
Average of dry and wet samples.....	2.46% MoS ₂
Content of molybdenite, using average analysis.....	2,081.16 lb.
Amount of concentrate obtained.....	2,135.0 lb.
Analysis of concentrate.....	93.44% MoS ₂
Content, 2,135 pounds, 93.44 per cent.....	1,994.94 lb.
Analysis of tailing, average of daily samples.....	0.12% MoS ₂
Calculated analysis of tailing by difference.....	0.12% MoS ₂
Recovery of molybdenite.....	95.8%

Procedure Followed.—The ore was crushed in jaw crusher to one inch and elevated to a storage bin from which it was fed through a Vezin sampler taking a one-twentieth cut. The ore after passing the sampler

went direct to a 4½-foot Hardinge mill operating in closed circuit with a simplex Dorr classifier. The classifier overflow went direct to two 1 by 6-foot, flat-bottom Callow roughing cells. The rougher concentrate was cleaned in two 1 by 3-foot cleaner cells operated in series. The cleaner tailing was returned to the head of the rougher cells by an air lift. The cleaner concentrate from the last cleaner cell was pumped to a Callow belt-screen equipped with an 80-mesh screen. The product remaining on the screen was the finished concentrate and was collected in a storage tank. The material passing through the screen was returned to the head of the rougher cells.

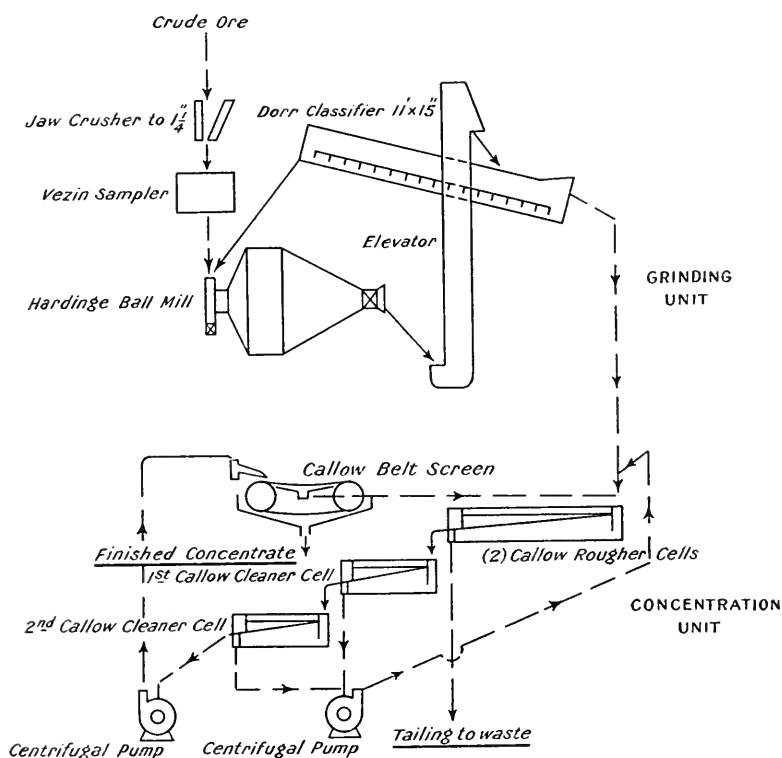


Figure 3. General flow-sheet for concentration of molybdenite ore.

NOTES ON FLOTATION

Considerable experimental work was done during the treatment of the first carload. The work was confined chiefly to the operation of the classifier and Callow belt-screen, and to the use of addition reagents such as lime and soda ash.

In order to produce a high-grade concentrate an endeavour was made to keep the molybdenite in the classifier overflow as coarse as possible, but at the same time free from any attached particles of gangue or other sulphide. In this connexion it was found necessary to crush until

the classifier overflow did not contain more than 1 per cent on a 35-mesh Tyler screen. The classifier was equipped with air and water sprays and the feed was allowed to plunge into the classifier so as to produce a boiling action in the settling-area of the classifier which tended to bring the flake to the surface. Water sprays were used on the rakes and caused considerable flake to be washed out of the oversize. An air spray at the water edge was used to blow the flake which had been washed out of the oversize over to the overflow as fast as it was washed down. This arrangement was found very satisfactory and reduced the quantity of flake returning to the mill by approximately 50 per cent.

The concentrate from the last cleaner cell was fed to a small 1-inch centrifugal pump together with a large volume of fresh water. The control of the quantity of water fed with the concentrate to the pump was found to be very important. Too much water broke up the flocculation of the molybdenite allowing too much molybdenite to pass through the Callow belt screen and be returned to the circuit. Too little water did not allow the molybdenite to flocculate free from the entangled pyrite and gangue, thus lowering the grade of the concentrate obtained on the screen. A series of analysis of the concentrate before and after passing over the screen showed that the use of the screen raised the grade 7 per cent to 8 per cent in MoS_2 content. The object of using the centrifugal pump was to re-flocculate the molybdenite in a pulp of clean water so that the pyrite and gangue would be delivered on the screen free from any entanglement in the flocculated molybdenite.

The oils used were kerosene and steam-distilled pine oil. The kerosene found to be most satisfactory was British American Oil Co's. "Lampolene" brand. The amount of kerosene used varied slightly from time to time, but at no time when a neutral pulp was used did it exceed 0.80 pound per ton. The quantity of pine oil used averaged only 0.4 pound per ton. The quantity of pine oil was found to affect the grade of the concentrate, too much oil lowering the grade. This effect was probably due to the well-known action of pine oil in a closed water circuit. Most of the selective power of pine oil is used up in its first contact with the ore. In this case practically all the pine oil in the flotation concentrate is returned back with the undersize from the screen. This building up of dead pine oil produced a voluminous froth which had very little carrying property for molybdenite. The result of this could be observed by watching the molybdenite gradually start to spread down the cells and instead of coming off right at the feed-end it comes off all over the cell. This building up of dead pine oil not only lowers the grade of the concentrate but also increases the loss in the tailing.

The addition of lime or soda ash to the pulp proved to be of no particular advantage. Lime was tried and the amount was varied from 1 to 5 pounds per ton of ore. A slight improvement was observed in the grade of the concentrate, but this was offset by a decided increased loss in the tailing. The increase in grade of the concentrate was about 1 per cent of MoS_2 and the increased loss in the tailing approximately 0.1 per cent. The same increase in the grade of the concentrate can be obtained in a neutral pulp with a corresponding tailing. The addition of lime gave a very different acting pulp from that obtained under neutral conditions; the quantity of kerosene could be doubled without any over-oiling effect

being observed in cells. It was necessary to use over $\frac{1}{4}$ pound per ton more kerosene than in a neutral pulp to keep the tailing clean. The lime apparently tended to emulsify the kerosene oil forming an emulsion which did not readily break down when coming in contact with the flakes of molybdenite. The increased loss in the tailing was from very fine flake only.

Soda ash had much the same effect on the oils, it being necessary to use a much larger amount of kerosene. The grade of concentrate was not improved as there was a tendency to float gangue material. The tailing was also higher.

An important and significant fact was observed in connexion with the operation of the ball mill. At one time during the run the characteristic froth in the rougher cells started to disappear and be replaced by a more voluminous heavy and slow-breaking froth. The molybdenite instead of appearing on the convexed face of the bubbles near their dome and all coming off in the first foot of the cells, appeared in dead-looking clots carried in the troughs between the contact of bubble surfaces and spreading down the whole length of the cell before coming off. The reagents were checked and found to be unchanged and the ore looked as clean as usual. The density of the pulp discharging from the mill was observed to be much thicker than usual. With the addition of more water to the ball mill the cells immediately cleaned up and operation came back to normal. During the time the pulp in the mill was thick there was considerably more loss of molybdenite in the tailing, and what is more important, the grade of the concentrate was considerably reduced due to the flotation of more iron. The action of the oils on this thick pulp in the ball mill was affected in some manner, entirely changing the characteristics displayed by a thinner pulp.

The density of the pulp in the ball mill during the run was between 45 per cent and 50 per cent solids, but when increased above 60 per cent poor results were obtained in the flotation cells. Any thickening of the pulp in the ball mill was indicated in the cells immediately by the characteristic froth referred to above.

SUMMARY AND CONCLUSIONS

The ore is very amenable to concentration by flotation. With proper manipulation and under the right conditions, as outlined above, a high-grade concentrate, 93 per cent MoS_2 , is obtained with recoveries in excess of 95 per cent of the molybdenite content in the ore. Concentrates assaying over 96 per cent MoS_2 were made during the runs under the best conditions. The concentrates are exceptionally free from deleterious substances; are suitable for the manufacture of molybdic acid, molybdic salts, molybdenum metal, ferro-molybdenum, and should command a high market price.

Report No. 216

EXAMINATION OF KIRKLAND LAKE MILL TAILINGS

C. S. Parsons

Shipment.—A shipment of 50 pounds of mill tailings from the Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ont., was received October 16, 1924.

Purpose of Experimental Tests.—The test work was conducted to determine whether the values in the tailings could be recovered by flotation, and also to get some idea of the reasons for the high tailing losses.

Assay of Sample.—The assay of the tailings submitted for test purposes showed them to contain 0.06 ounce gold per ton. It is reported that at times the tailings exceed this value, depending on the refractoriness of the ore being treated.

Experimental Tests.—The experimental tests consisted of screen analysis of the tailings, classification tests, and flotation tests.

Screen Analysis of Tailings from Wet Sizing Test on Tyler Standard Screens.

Head sample: 0.06 ounce gold per ton.

Mesh	Weight per cent	Assay Au oz./ton	Per cent of values
—100+150.....	19	0.10	35
—150+200.....	12	0.07	15
—200.....	69	0.04	50
Totals.....	100	0.055	100

The above screen test shows that 31 per cent of the tailing stays on a 200-mesh screen; 69 per cent passes a 200-mesh screen; that this 31 per cent oversize contains 50 per cent of the gold values in the tailings.

Classification Tests on Tailing.—Several classification tests were made to determine the distribution of the values in the classified products, sands and slimes. The results of a number of these tests are given below:—

Product	Weight per cent	Assay Au oz./ton	Per cent of values
Sands.....	76.5	0.08	89.7
Slimes.....	23.5	0.03	10.3
Totals.....	100.0	0.068	100.0
Sands.....	61.8	0.10	80.2
Slimes.....	38.2	0.04	19.8
Totals.....	100.0	0.077	100.0
Sands.....	80.7	0.07	93.5
Slimes.....	19.3	0.02	6.5
Totals.....	100.0	0.06	100.0
Sands.....	73.3	0.08	95.6
Slimes.....	26.7	0.01	4.4
Totals.....	100.0	0.061	100.0

The above classification tests show that the high tailing values are in the coarser material and that cyanidation has extracted the values to a much greater extent from the fine slimes.

Flotation Tests on Classified Sands.—The following flotation tests were made on the sand products from the classification tests given above. The values in the flotation products do not check up with the values in the classified sands, probably due to the small amount of material being used in the tests, the low assay value of the material and products, and the difficulty in cleaning up the testing apparatus.

Heads: Classified sand—0.10 ounce per ton

Product	Weight %	Assay Au oz./ton	Per cent of values
Concentrate.....	7.2	0.40	60.6
Tailing.....	92.8	0.02	39.4
Totals.....	100.0	0.047	100.0
Concentrate.....	3.0	0.70	44.5
Middling.....	8.5	0.10	18.0
Tailing.....	88.5	0.02	37.5
Totals.....	100.0	0.047	100.0

Heads: Classified sand—0.08 ounce per ton

Concentrate.....	4.1	0.60	44.1
Middling.....	13.0	0.05	11.5
Tailing.....	82.9	0.03	44.4
Totals.....	100.0	0.056	100.0

The flotation results on the classified sand tailings are, so far, not very promising. They show a possible recovery of 40 per cent of the values in the cyanide tailings, in a concentrate assaying 0.6 ounce per ton or \$12 per ton. This concentrate would represent about 4 per cent by weight of the ore fed to the mill.

SUMMARY

From the results of the screen test and classification tests given above it would seem that coarse grinding is responsible, to a large extent, for the high tailing losses. Cyanidation seems to have given good extractions on the -200-mesh material and on the classified slimes. Fifty per cent of the tailing values is in the 31 per cent of the tailings remaining on 200-mesh. Over 80 per cent of the tailing values is in the sands from classification, and less than 20 per cent in the slimes. An all-slime process should, therefore, increase the recovery materially and give a low tailing. It would seem that more efficient classification is required in the tube mill circuit. This may mean the installation of a bowl to receive the classifier overflow, and of an additional tube mill in circuit with the bowl, in order to make an all-slime process. The problem is whether the cost of this additional installation would be offset by the increased recoveries.

It is the opinion of many engineers that it is the presence of tellurides which accounts for the high tailing losses. If it is the case that tellurides grind readily, one would expect them to slime easily, and that the slimes would carry the higher values. This is not the case, and would

lead one to believe that the losses are in the coarse pyrite. A microscopic examination is being made of the coarse pyritic concentrates in the tailings to determine if the tellurides are present with the pyrite. Flotation of cyanide tailings does not seem to work very satisfactorily. This was found to be the case when tried in the Cobalt camp.

Report No. 217

CONCENTRATION OF A SILVER-LEAD ORE FROM PORTLAND CANAL DISTRICT, STEWART, BRITISH COLUMBIA

C. S. Parsons

Shipment.—A shipment of 54 pounds of silver-lead ore was received August 26, 1924, from S. G. O. Chalmers, Stewart, B.C.

Location.—The shipment was made from the E. G. group of mineral claims situated on the Canadian side of Portland canal, five miles south of Stewart, and was supposed to be representative of the ore from these claims.

Characteristics of the Ore.—The ore is a complex mixture of lead, zinc, copper, and iron sulphides, carrying silver and gold values. The silver values predominate. The gangue is of a siliceous nature.

Sampling and Analysis.—The shipment was sampled and found to contain the following on analysis:—

Lead.....	2.48 per cent	Iron.....	4.00 per cent
Zinc.....	1.60 "	Gold.....	0.05 oz. per ton
Copper.....	0.30 "	Silver.....	37.00 "

Purpose of Experimental Tests.—Experimental work was desired to determine methods of treatment to recover the values in the ore.

Scope of Experimental Work.—Methods of treatment outlined in the following description of the experimental work include: First, fine grinding followed by flotation of all the sulphides to obtain a concentrate carrying the precious metal values which could be economically marketed to the smelters. Second, fine grinding followed by selective flotation to obtain a lead concentrate carrying part of the precious metals, and a zinc concentrate carrying the remainder of the precious metal values. Third, the cyanidation of the flotation tailings to obtain further recoveries of the precious metal values.

Experimental Tests.—The results of the experimental tests are given in tabulated form following:—

TESTS Nos. 1, 2, AND 3

FLOTATION OF ALL THE SULPHIDE MINERALS

This series of flotation tests was made to compare the results obtained in a lime, soda ash, and neutral pulp.

Test No. 1

Flotation was made in an alkaline pulp produced by the addition of lime. The ore was crushed in a ball mill to pass a 65-mesh screen. A

screen test showed that 60 per cent passed through 200 mesh. The following reagents were added during the grinding:—

Lime.....	5.0 pounds per ton
Mixture of 50 per cent gas tar.....	} 0.5 pound per ton
50 per cent coal-tar creosote.....	

A small amount of steam-distilled pine oil for frothing was added to flotation cell.

Test No. 2

The flotation was made in an alkaline pulp produced by the addition of soda ash. The same procedure and oils were used as in test No. 1. The flotation tailing was tabled.

Test No. 3

The flotation was made in a neutral pulp. The same procedure and oils were used as in tests Nos. 1 and 2. The flotation tailing was tabled.

Conclusions from Tests Nos. 1, 2, and 3.—As expected, the lime pulp caused the iron sulphides to report in the tailing and also interfered with the flotation. Using the soda ash pulp a high recovery of the zinc and lead was obtained, and the greater part of the iron sulphides reported with the lead and zinc concentrates. It will be noted that a much higher recovery of the silver and gold values was obtained than in test No. 1. The recoveries of lead and zinc were not so high in test No. 3 in which a neutral pulp was used, and a larger proportion of the iron sulphides reported in the tailing than in test No. 2. The results of the table tests on the flotation tailing of these three tests give some idea of the association of the values in the flotation tailing. One point of importance to note is that there is more silver in the tailing than there is combined zinc and copper minerals to account for its association being entirely with those minerals.

TESTS Nos. 4, 7, 9, AND 11, SELECTIVE FLOTATION TESTS

This is a series of selective flotation tests made to produce a lead concentrate and a zinc concentrate which could be marketed separately. No attempt was made to reclean the lead and zinc concentrates.

Test No. 4

The ore was ground in a ball mill to pass a 65-mesh screen. The following reagents were used.

Lead reagents—

5.0 pounds per ton	soda ash.....	added to ball mill.
0.15	"	Z eake.....
0.30	"	sodium cyanide.....
0.05	"	steam-distilled pine oil added to cells.

Zinc reagents—

1.0	"	copper sulphate.
0.25	"	XY mixture.
0.25	"	No. 2 neutral creosote oil from Dominion Tar and Chemical Co.

Test No. 7

The same procedure was followed as in test No. 4.

Lead reagents—		
5.0	pounds per ton	soda ash.
0.15	"	Z cake.
0.30	"	sodium cyanide.
0.05	"	pine oil.
Zinc reagents—		
1.0	"	copper sulphate.
0.25	"	YZ mixture.
0.25	"	No. 2 neutral creosote.

Test No. 9

The ore was ground as in the above tests.

Lead reagents—		
5.0	pounds per ton	soda ash.
0.01	"	QED
0.3	"	sodium cyanide.
0.05	"	pine oil.
Zinc reagents—		
1.0	"	copper sulphate.
0.25	"	YZ mixture.
0.25	"	No. 2 neutral creosote.

Test No. 11

The same procedure was followed as in the previous tests.

Lead reagents—		
4.0	pounds per ton	soda ash.
0.25	"	mixture of water-gas tar 50 per cent, coal-tar creosote 50 per cent.
0.2	"	sodium cyanide.
0.05	"	creylic acid.
Zinc reagents—		
1.0	"	copper sulphate.
0.3	"	Barrett's No. 634.
0.05	"	pine oil.

Conclusions from Tests 4, 7, 9, and 11.—Tests Nos. 7 and 11 gave the best results. Test No. 4 was also good, the same reagents being used as in test No. 7. In test No. 9 for some unaccountable reason the lead and silver values were much lower than the head sample so that no definite conclusions can be drawn. However, there seems to be no advantage in using reagent QED in place of Z cake for the flotation of the lead. In test No. 11 oils were used to float the lead in place of Z cake, and a better selective action was obtained between the lead and zinc. Special note should be taken when comparing the grade of the lead and zinc concentrates in these tests that they were not recleaned. In practice these concentrates would be recleaned and the grade considerably improved. There is little doubt that a 50 per cent lead concentrate and a 40 per cent zinc concentrate could be obtained.

TESTS Nos. 5, 6, AND 8, FLOTATION TESTS WITH ADDITION OF SODIUM SULPHIDE

The following is a series of tests made to determine whether a high recovery of the silver values could be obtained by the addition of sodium sulphide.

Test No. 5

The ore was ground in a ball mill to pass 60 mesh with 5 pounds per ton sodium sulphide, and an attempt was made to obtain a product containing the silver values.

Reagents—
 5.0 pounds per ton sodium sulphide (Canadian Electro-Products).
 0.2 " Z cake.

It was noted that very little zinc was floating, so 0.2 pound per ton YZ mixture was added to bring it up.

Test No. 6

Same procedure was followed as in test No. 5, but in this test selective flotation of the lead and zinc was attempted.

Lead reagents—
 5.0 pounds per ton sodium sulphide to ball mill.
 0.2 " Z cake.....to ball mill.
 0.3 " sodium cyanide to cells.
 0.05 " pine oilto cells.

Zinc reagents—
 1.0 " copper sulphate.
 0.5 " No. 2 neutral creosote oil.

Concentrates 1 and 2 were taken off with the reagents used for the flotation of the lead and in the presence of cyanide. Concentrate 3 was taken off by the reagents used for the flotation of zinc after addition of copper sulphate.

Test No. 8

The procedure was that followed in tests Nos. 5 and 6. Selective flotation of the lead and zinc was not attempted.

Reagents—
 0.75 pound per ton mixture of 60 per cent coal-tar creosote, 40 per cent coal tar.
 0.05 " " steam-distilled pine oil.

Conclusions from Tests 5, 6 and 8.—The sodium sulphide instead of increasing the recovery of the silver had the effect of increasing the loss in the tailing. Comparing tests Nos. 5 and 8 it will be seen that although the grade of the concentrates differ, the recoveries are practically the same. The difference between the grade of concentrate is due to the use of oils in test No. 8 instead of Z cake as in test No. 5. Test No. 6 in which an attempt was made to selectively float a lead and zinc concentrate is worthy of careful consideration, as it throws considerable light on the association of the silver values. According to the results obtained in this test, at least a considerable portion of the silver must be present as a separate mineral and not associated with the other sulphides as it will be observed that the iron sulphides collected in No. 1 concentrate, but that only 5.5 per cent of the silver values reported there. The second concentrate consisted chiefly of lead values but they contained 18.3 per cent of the total silver, whereas the third concentrate contained 55 per cent of the silver values and assayed 344.8 ounces per ton.

Test No. 10—Table Concentration

This test was made on a small-size Wilfley table. A fair grade of concentrate was obtained, but the recovery of both the lead and silver values was poor.

TESTS Nos. 12 AND 13, FLOTATION TESTS WITH FINE GRINDING

These two tests were run to determine if the recovery of the silver could be increased by finer grinding.

Test No. 12

This is a selective flotation test for the production of a lead and zinc concentrate. The ore was ground to pass 100 mesh. The method of conducting the test was the same as in the previous selective tests:—

Lead reagents—

5.0 pounds per ton soda ash.
0.01 " QED
0.3 " sodium cyanide.

A lead concentrate was made and was re-cleaned, producing a lead middling product.

Zinc reagents—

1.0 pounds per ton copper sulphate.
0.2 " YZ mixture.
0.4 " No. 2 neutral creosote.

The concentrate was not re-cleaned as the amount was too small to work with.

Test No. 13

This was a straight flotation test to produce a concentrate containing the lead and zinc together with the silver and gold values:—

Reagents: 0.4 pound per ton coal-tar creosote (Dominion Tar & Chemical Co.)
0.4 " No. 2 neutral creosote "

The acid creosote oil was used to float the lead, and the neutral creosote oil to float the zinc.

Conclusions from Tests Nos. 12 and 13.—Finer grinding did not improve the recovery of the silver and gold values in these tests, but the grinding may not have been carried far enough, because a screen test made on the tailing, given later, shows that the silver values are higher in the coarser products.

CYANIDE TESTS

J. S. Godard

Cyanidation of the Ore.—This series of tests was made on the crude ore ground to pass 100 mesh. The time of treatment was the only variable in this series.

Time: No. 1, 24 hours.	No. 2, 30 hours.	No. 3, 48 hours.
Details: 200 grammes ore, -150 mesh.		
600 " water.		
0.68 " sodium cyanide, 0.15 per cent solution.		
0.5 " lime, 5 pounds per ton.		

The above charge was used in each test.

Results:—

Test No.	Assay Silver oz/ton		Extraction per cent	Consumption lbs./ton	
	Head	Tailing		Cyanide	Lime
1.....	37.09	22.19	40.2	5.60	2.90
2.....	37.09	22.09	40.4	4.92	3.05
3.....	37.09	21.80	41.5	4.74	3.25

Cyanide Tests Nos. 4, 5, and 6

Cyanidation of Flotation Tailing.—This series of tests was made on flotation tailing from test No. 11. The strength of solution was varied, but time kept constant at 48 hours. Strength of solution: No. 4, 0.05 per cent; No. 5, 0.10 per cent; No. 6, 0.20 per cent.

Time, 48 hours.

Details: 200 grammes ore, -150 mesh.
600 " water.

Results:—

Test No.	Ag. oz/ton		Au. oz/ton		Extraction Ag. per cent	Consumption lbs./ton	
	Heads	Tailing	Heads	Tailing		Cyanide	Lime
4.....	9.56	4.54	0.02	trace	52.5	1.7	1.8
5.....	9.56	4.08	0.02	trace	57.2	1.8	1.7
6.....	9.56	3.88	0.02	trace	59.4	2.8	1.5

Conclusions from Cyanide Tests Nos. 1-6.—The cyanidation of the crude ore was not successful, less than 50 per cent of the silver being extracted. The cyanidation of the flotation tailing was more successful from an extraction point of view. Over 50 per cent of the silver in the tailing was extracted, which in test No. 6 is 5.68 ounces per ton, or \$3.86, with silver at 68 cents per ounce.

General Summary and Conclusions.—The following screen test was made on the flotation tailing from test No. 4 for the purpose of obtaining information on the condition of the remaining silver.

Screen Analysis on Tailing Test No. 4

Heads: Lead, 0.17%. Zinc, 0.18%. Silver, 7.68 oz/ton. Gold, 0.02 oz/ton.

Mesh	Weight		Analysis				Per cent of silver values
	Grms.	%	Silver	Gold	Lead	Zinc	
			oz/ton	oz/ton	%	%	
+65	3.5	0.5	17.12	0.13	0.1
+100	54.2	6.9	13.2	0.04	0.13	0.33	11.4
+150	132.5	16.9	11.03	0.03	0.18	0.32	23.3
+200	119.5	15.3	9.96	0.02	0.15	0.31	19.1
-200	471.5	60.4	6.11	0.01	0.16	0.10	46.1

The screen test indicated that the ore should be ground to at least 150-mesh.

The table tests made on the flotation tailing from tests Nos. 1, 2, and 3 showed that about 40 per cent of the silver remaining in the flotation tailing could be recovered, but the concentrate produced was rather low grade. It was necessary to cut considerable gangue with concentrate because the amount of concentrate in the feed was so small. In actual practice on a full-sized table a higher grade concentrate could be produced. The suggestion is made that this table concentrate, if not sufficiently high grade to ship, might be returned and reground with the feed, and pass again through the mill circuit.

There was not sufficient ore to continue the flotation tests to show which of the two methods of flotation would be the more satisfactory. The comparison of the two systems depends on the possibility of producing both a lead and zinc concentrate of marketable grade by selective flotation, and comparing the combined net value of the concentrate against the net value of a concentrate produced by straight flotation containing zinc in amounts which would subject it to heavy penalties by the smelter.

Cyanidation of the crude ore was not successful, only 40 per cent of the silver being extracted. Cyanidation of the flotation tailing gave a little higher recovery, approximating about 55 per cent of the silver in the flotation tailing. There was not sufficient ore to run a cumulative cyanide test to determine the tendency of the solutions to become fouled. It is possible that considerable fouling of the solution would take place which would result in increased cyanide consumption and precipitation difficulties. This test should be run before drawing any definite conclusions as to the advisability of cyaniding.

The series of tests conducted on this sample are purely of a preliminary nature, and it will be necessary to do more detailed work in order to determine the best flow-sheet for the ore. A sample of at least 500 pounds would be required for this work. It should represent accurately the milling grade of ore in the mine, and should be a fresh sample taken from all working-faces in proportion to the amount of material which will be mined from each face.

Results of Experimental Tests

Test No.	Product	Weight		Analysis						Per cent of values			
		Grms.	%	Lead %	Zinc %	Copper %	Iron %	Silver oz./ton	Gold oz./ton	Lead	Zinc	Silver	Gold
1	<i>Flotation of all the sulphides—</i>												
	Concentrate.....	97.2	9.7	23.7	9.33	259.65	0.35	91.1	54.7	65.8	48.6
2	Tailing.....	900.0	90.2	0.25	0.83	14.50	0.04	8.9	45.3	34.2	51.4
	Concentrate.....	124.7	12.5	18.7	12.37	275.45	0.35	94.7	89.8	86.6	71.0
3	Tailing.....	873.0	87.5	0.15	0.20	6.04	0.02	5.3	10.2	13.4	29.0
	Concentrate.....	104.0	10.4	21.30	11.46	204.35	0.35	89.2	71.5	74.1	57.2
1	Tailing.....	897.5	89.6	0.30	0.53	10.73	0.03	10.8	28.5	25.9	42.8
	<i>Table tests on flotation tailings—</i>												
1	Table concentrate.....	42.0	8.1	2.56	9.43	94.68	0.20	58.5	82.2	43.2	30.2
	Table tailing.....	478.0	91.9	0.16	0.18	11.10	0.04	41.5	17.8	56.8	69.8
2	Table concentrate.....	67.0	13.8	0.42	0.8	18.08	0.04	40.3	29.7	30.2	22.7
	Table tailing.....	419.0	86.2	0.10	0.03	6.70	0.02	59.7	70.3	69.8	77.3
3	Table concentrate.....	54.0	12.9	1.13	4.05	40.8	0.08	56.3	77.9	39.3	27.8
	Table tailing.....	363.0	87.1	0.13	0.17	9.33	0.03	43.6	22.1	60.7	72.2
4	<i>Selective flotation tests—</i>												
	Lead concentrate.....	86.0	8.6	25.30	10.90	2.93	14.18	347.4	0.40	91.0	61.4	78.1	62.5
4	Zinc concentrate.....	28.5	2.8	2.29	15.10	0.62	13.79	53.42	0.10	2.7	28.1	4.0	5.2
	Tailing.....	890.0	88.6	0.17	0.18	0.05	2.35	7.68	0.02	6.3	10.5	17.9	32.3
7	Lead concentrate.....	93.0	9.2	25.24	10.42	2.09	13.99	316.0	0.40	91.4	62.1	76.5	75.2
	Zinc concentrate.....	26.0	2.6	1.33	21.49	0.43	10.84	66.35	0.15	1.4	36.2	4.5	8.0
7	Tailing.....	884.0	87.2	0.21	0.03	0.03	2.18	8.29	0.01	7.2	1.7	19.0	17.8
	Lead concentrate.....	41.4	4.1	24.35	10.75	2.70	12.51	304.4	0.40	76.2	26.5	43.2	38.5
7	Zinc concentrate.....	87.5	8.7	1.97	13.97	0.83	18.36	109.0	0.20	13.1	73.0	32.7	41.0
	Tailing.....	878.0	87.2	0.16	0.01	0.06	2.19	7.99	0.01	10.7	0.5	24.1	20.5
7	Lead concentrate.....	58.5	5.3	37.62	7.23	439.35	0.60	85.3	26.0	63.1	53.1
	Zinc concentrate.....	56.0	5.1	3.07	20.43	99.00	0.20	6.7	70.9	13.7	17.1
7	Tailing.....	987.0	89.6	0.21	0.05	9.56	0.02	8.0	3.1	23.2	29.8

Results of Experimental Tests—Concluded

Test No.	Product	Weight		Analysis							Per cent of values			
		Grms.	%	Lead %	Zinc %	Copper %	Iron %	Silver oz./ton	Gold oz./ton	Lead	Zinc	Silver	Gold	
5	<i>Flotation tests with addition of sodium sulphide—</i> Concentrate..... Tailing.....	146.0	14.6	13.55	3.74	1.14	10.87	162.0	0.20	84.0	38.3	62.8	46.0	
		856.5	85.4	0.44	1.03	0.12	2.30	16.44	0.04	16.0	61.7	37.2	54.0	
6	Concentrate No. 1.....	39.7	4.0	47.3	1.54	0.24	15.86	53.30	0.10	7.7	4.1	5.5	6.5	
	Concentrate No. 2.....	40.7	4.1	38.55	1.74	0.78	8.70	171.50	0.20	64.1	4.7	18.3	12.9	
	Concentrate No. 3.....	60.5	6.1	7.89	8.11	1.35	9.65	344.80	0.40	19.5	32.7	55.0	38.7	
	Tailing.....	851.5	85.8	0.25	1.03	0.06	2.16	9.40	0.03	8.7	58.5	21.2	41.9	
8	Concentrate.....	49.0	5.2	35.0	5.31	2.65	17.63	316.36	0.44	84.6	81.0	54.5	62.3	
	Tailing.....	899.0	94.8	0.35	1.26	0.08	2.21	14.46	0.04	15.4	19.0	45.5	37.7	
10	<i>Table concentration—</i>													
	Concentrate.....	96.7	3.3	40.72	8.98	1.49	15.28	228.74	0.30	57.2	22.2	19.8	13.9	
	Middling.....	141.0	4.8	2.51	12.52	0.83	9.97	64.87	0.15	5.1	45.2	8.2	9.7	
	Tailing.....	2,161.0	74.4	0.56	0.36	0.15	2.41	24.51	0.05	17.7	20.1	47.7	51.4	
	Slimes.....	510.2	17.5	2.68	0.95	0.40	3.31	53.10	0.10	20.0	12.5	24.3	25.0	
12	<i>Flotation tests—fine grinding—</i>													
	Lead concentrate.....	57.5	5.8	38.18	11.19	460.7	0.60	87.9	38.9	69.2	59.1	
	Lead middling.....	56.7	5.7	2.00	4.51	50.0	0.10	4.4	15.1	7.5	9.8	
	Zinc concentrate.....	34.7	3.5	2.40	20.38	77.85	0.15	3.2	43.4	7.2	9.0	
	Tailing.....	851.1	85.1	0.13	0.04	7.16	0.015	4.4	2.1	16.1	22.1	
13	Concentrate.....	142.0	14.2	18.14	10.99	230.30	0.30	94.5	98.1	83.7	71.2	
	Tailing.....	858.0	85.8	0.17	0.04	7.44	0.02	5.5	1.9	16.3	28.8	

Report No. 218

CONCENTRATION OF TITANITE (SPHENE) FROM PEGMATITE

C. S. Parsons

Shipment.—A shipment of 42 pounds of pegmatite rock containing titanite, or sphene, was submitted by Messrs. Spence and Cole of the Mines Branch, Ottawa.

Source of Shipment.—The shipment was obtained from a deposit located on lot 3, concession III, Alleyn township, Pontiac district, Quebec.

Analysis.—The sample assayed 5.3 per cent titanium oxide (TiO_2) contained in the mineral sphene.

Character of the Rock.—The sphene was associated with a considerable quantity of dark green hornblende and some calcite in a quartz pegmatite.

EXPERIMENTAL TESTS

General Considerations.—The rock was tough and difficult to crush. Crushing costs would be the chief factor in the treatment. The reduction of the rock was therefore not carried any further than necessary, only to the point of separation of the sphene from the quartz-calcite-hornblende gangue. The sphene is more brittle than the gangue and consequently crushes finer, but in parting from the gangue especially the quartz, a thin shell of sphene is left attached. Fine grinding would be required to free this adhering shell, but this operation would not be economical. In view of the above considerations, the procedure followed was to rough out a clean tailing at as coarse a size as possible, so that any recrushing required to obtain a cleaner product would be done on a smaller amount of material.

General Description of Tests.—By crushing a sample to different degrees of fineness and carefully examining the sizes, it was determined that -3 mesh was the coarsest size from which there was any likelihood of obtaining a sufficiently clean tailing. The size $-3+10$ was jigged, and a final tailing and rougher concentrate made. The concentrate contained the greater part of the hornblende which has practically the same specific gravity as sphene. Considerable other gangue was also present. A concentrate and tailing were made from this $-3+10$ size. The concentrate was recrushed to 20 mesh and re-treated as described later. The $-10+20$ -mesh material was jigged, and only a concentrate and final tailing were made. This concentrate was also recrushed. The -20 -mesh material was tabled without further sizing. A final table tailing was made and a table concentrate which was added to the recrushed concentrate from the jig. All the recrushed concentrate was sized and recleaned on tables producing a sphene-hornblende product and a secondary tailing which was not further treated. The sphene-hornblende product was fed to an Ulrich magnetic separator and the hornblende was removed, leaving a sphene concentrate containing some quartz.

Results of Tests.—The weights of the sizes obtained by crushing to 3 mesh were as follows:—

Total weight of sample.....	42 lbs.
Assay.....	5.03 % TiO_2
Content.....	2.1126 lbs. TiO_2
-3+10 mesh weight.....	30 lbs.
Per cent total weight.....	71.44
-10+20 mesh weight.....	6 lbs.
Per cent total weight.....	14.28
-20 mesh weight.....	6 lbs.
Per cent total weight.....	14.28

Results of Jigging -3+10-mesh Material.—

Tailing	Weight.....	19 lbs.
	Assay.....	0.90% TiO_2
	Content.....	0.152 lbs. TiO_2

Summary.—45.2 per cent of the original sample of 42 pounds was eliminated as a final tailing containing 7.2 per cent of the total titanium oxide (TiO_2)

Results of Jigging -10+20-mesh Material.—

Tailing	Weight.....	3.48 lbs.
	Assay.....	0.50% TiO_2
	Content.....	0.0174 lbs. TiO_2

Summary.—8.28 per cent of the original sample of 42 pounds was eliminated as a final tailing, containing 0.82 per cent of total titanium oxide (TiO_2).

Results of Table Concentration -20-mesh Material.—

Tailing	Weight.....	2.88 lbs.
	Assay.....	1.0% TiO_2
	Content.....	0.0288 lbs. TiO_2

Summary.—6.85 per cent of the original sample was eliminated as a final tailing containing 1.365 per cent of the total TiO_2 .

Recapitulation of Results

Product	Weight		Analysis TiO_2 %	Content lbs. TiO_2	Per cent of values
	lbs.	%			
Final concentrate.....	4.00	10.0	29.84	1.20	65.3
Final middling.....	1.75	4.4	7.61	0.133	7.3
Final tailing.....	25.50	63.7	0.60	0.196	10.7
Final secondary tailing.....	2.50	6.3	2.0	0.05	2.7
Final hornblende product.....	4.25	10.6	1.1	0.046	2.5
-100 mesh product.....	2.00	5.0	10.58	0.212	11.5
Total.....	40.00	100.0	4.6	1.837	100.0

Summary.—The results of the above concentration tests show a concentration ratio of 10 of rock to 1 of concentrate. The concentrates averaged 30 per cent TiO_2 with a recovery of 65 per cent of the titanium content of the rock. The recovery could probably be increased to 70 or 75 per cent by further treatment of the middling product, and concentration of the fines.

Report No. 219

EXPERIMENTAL TESTS ON CADWALLADER CREEK MILL TAILINGS

J. S. Godard

Shipment.—A shipment of two bags weighing 96 pounds, was received on October 13, 1924, from A. W. Davis, Resident Engineer, Kamloops, B.C. One bag was marked "Coronation mill tailings" and weighed 31 pounds; the other "Lorne mill tailings" and weighed 65 pounds.

Purpose of Experimental Tests.—Experimental tests were conducted to determine a method for the recovery of the values remaining in the tailings.

Sampling and Analysis.—A sample was taken from each lot of the tailings as received, using a riffle sampler. Assays of the samples showed them to contain:—

Coronation Tailing—

Gold.....	0.30 ounce per ton
Silver.....	0.08 " "
Arsenic.....	0.38 per cent
Iron.....	3.71 " "

Lorne Tailing—

Gold.....	0.08 ounce per ton
Silver.....	" "
Arsenic.....	0.43 per cent
Iron.....	4.19 " "

Experimental Tests.—Table concentration, amalgamation, and cyanide tests were made to determine the recovery of the values by these methods.

CORONATION TAILING

Table Concentration Test.—A 3,000-gramme sample of the tailing as received, which showed by screen tests to be 48-mesh material was concentrated on a small Wilfley table with the following results:—

Product	Weight per cent	Gold oz/ton	Per cent of values
Concentrate.....	0.90	3.56	16.7
Middling.....	1.10	0.20	1.1
Tailing.....	98.00	0.16	82.2

Amalgamation Test.—A 1,000-gramme sample of the tailing as received was amalgamated for two hours in a pebble jar using 10 per cent by weight of mercury and a pulp density of 1:1. A screen analysis was made of the amalgamation tailing which showed the following results:—

Product	Weight per cent	Gold oz/ton	Per cent of values
+48 }.....	8.8	0.10	3.6
—48+65 }.....			
—65+100.....	19.3	0.10	8.0
—100+150.....	25.4	0.14	14.9
—150+200.....	11.4	0.20	9.5
—200.....	35.1	0.44	64.0

Head sample.....	0.30 ounce per ton
Amalgamation tailing.....	0.24 " "
Recovery by amalgamation.....	20 per cent

Cyanidation Tests.—A series of bottle-type agitation cyanide tests was made, the results of which are given in the following table:—

Test No.	4	5	6	7
Weight, grms.....	300	300	300	300
Mesh.....	48	48	48	—150
Water, c.c.....	300	600	900	900
Potassium cyanide, per cent.....	0.2	0.2	0.2	0.2
Lime, pounds per ton.....	4	4	4	4
Heads, gold, oz/ton.....	0.30	0.30	0.30	0.30
Tailing, ".....	0.06	0.04	0.04	0.03
Recovery per cent.....	80.0	86.7	86.7	90.0
Cyanide consumption, pounds per ton.....	0.62	0.73	0.42	2.4
Lime consumption, pounds per ton.....	3.4	2.9	2.7	3.4
Time, hours.....	24	24	24	24

Summary of Tests on Coronation Tailing.—16.7 per cent of the gold values was recovered in a table concentrate representing by weight 1 per cent of the feed and assaying 3.56 ounces per ton gold, 15.52 per cent arsenic, 24.96 per cent iron. Twenty per cent of the gold values was recovered by amalgamation. The test shows that the greater proportion of the values is in the —200-mesh material and is not recoverable by amalgamation. Ninety per cent of the gold values was recovered by cyanidation from the tailing ground to 150 mesh, using a pulp dilution of 3:1 and 0.20 per cent potassium cyanide solution. Eighty-seven per cent of the values was recovered without further grinding and with a low cyanide consumption.

LORNE TAILING

Table Concentration Test.—A 2,655-gramme sample of the tailing as received, which showed by screen test to be 28-mesh material, was concentrated on a small Wilfley table with the following results:—

Product	Weight per cent	Gold oz/ton	Per cent of values
Concentrate.....	2.8	1.12	51.6
Middling.....	15.8	0.04	9.7
Tailing.....	81.4	0.03	38.7

Amalgamation Test.—A 1042-gramme sample of the tailing as received was amalgamated in a pebble jar for two hours, using 100 grammes of mercury and a pulp density of 1:1. A screen analysis of the amalgamation tailing gave the following:—

Product	Weight per cent	Gold oz/ton	Per cent of values
+48 mesh.....	22.7	0.04	16.2
—48+65 mesh.....	10.1	0.04	7.1
—65+100 mesh.....	25.9	0.02	9.3
—100+150 mesh.....	10.9	0.06	11.6
—150+200 mesh.....	8.4	0.06	8.9
—200 mesh.....	22.0	0.12	46.9
Head sample.....		0.08 ounce per ton	
Amalgamation tailing.....		0.056 " "	
Recovery by amalgamation.....		30.0 per cent	

Cyanidation Tests.—A series of bottle-type agitation cyanide tests was made, the results of which are given in the following table:—

Test No.	4	5	6
Weight, grms.....	290	309	305
Mesh.....	—150	28	28
Water, c.c.	870	927	915
Potassium cyanide, per cent.....	0.20	0.20	0.30
Lime, pounds per ton.....	4	4	4
Heads, gold, oz/ton.....	0.08	0.08	0.08
Tailing, “.....	0.04	0.02	0.03
Recovery, per cent.....	50	75	62.5
Cyanide consumption, pounds per ton.....	2.3	0.71	1.10
Lime consumption, pounds per ton.....	2.7	2.5	1.9

Summary of Tests on Lorne Tailings.—51.6 per cent of the gold values was recovered in a table concentrate representing by weight 28 per cent of the feed and assaying 1.12 ounces per ton gold, 8.12 per cent arsenic, 19.79 per cent iron. Thirty per cent of the gold values was recovered by amalgamation. The test shows that about 50 per cent of the values is in the —200-mesh material on which a poor recovery is made by amalgamation. Seventy-five per cent of the gold values was recovered by cyanidation of the tailings as received, using 0.20 per cent potassium cyanide solution. The cyanide consumption was less than one pound per ton.

CONCLUSIONS

The tests conducted on the mill tailings show that cyanidation gives the highest recovery of the values. As cyanidation was used for the recovery of the values from the Cadwallader Creek ore, it would seem that increased recoveries could be expected by more careful attention to the details of this practice, especially in the case of the Coronation ore.

Report No. 220

CONCENTRATION OF THE LEAD-ZINC ORE OF THE READER MINE, CALUMET ISLAND, QUEBEC

W. B. Timm¹ and C. S. Parsons

The favourable position of lead in the metal markets of the world has resulted in an active search for new lead properties, and the re-opening of idle properties. Hence a visit was made by the writer on November 13, 1924, to the Reader mine on Calumet island.

Situation.—The property comprises lots 9, 10, 11, and 12, range IV, township of Calumet, district of Pontiac in the province of Quebec. It is situated in the southern part of the island, one-half mile from the Roche Fendue channel of the Ottawa river, three miles from the village of Bryson and the recent power development at Calumet falls, and six miles from Campbells Bay on the Ottawa-Waltham branch of the Canadian Pacific railway.

¹ Introductory by W. B. Timm.

Operation.—The property was operated intermittently from 1896 to 1913 by the Grand Calumet Mining Co., Ltd., the Calumet Metals Co., and the Calumet Zinc and Lead Co. In 1898, 1,100 tons of high-grade ore was shipped to Belgium. In 1910 a small concentrating plant was built and trial shipments of lead and zinc concentrates were made to the Balbach smelter, Newark, N.J. In 1912 a 150-ton concentrator was built, equipped with jaw crusher, coarse rolls, screens, fine rolls, jigs, Huntingdon mill, Wilfley and Overstrom tables. This mill after being operated for only a few months, was closed down and was later destroyed by fire.

Occurrence.—There are two main zones, striking northwest, 1,400 feet apart at the southern exposures and 630 feet apart at the northern exposures. The western zone was opened up by prospect pits and shafts for 2,100 feet along the strike, and the eastern zone for 1,800 feet along the strike.

Development.—Development consists of four shafts varying in depth from 74 feet to 146 feet; several smaller shafts varying in depth from 17 feet to 35 feet, and numerous prospect pits. The total amount of drifting is approximately 500 feet.

Concentration Products from Milling Operations.—A marketable lead product was made containing on the average 65 per cent lead, 8 per cent zinc, 0.5 per cent copper, and 87 ounces of silver per ton. A low-grade zinc product was obtained containing on the average 3.5 per cent lead, 28 per cent zinc, 18 ounces silver per ton, and of high iron content. The tailing losses were high. Before the application of oil flotation, lead-zinc-silver ores possessing the characteristics of this ore presented a difficult problem in the matter of their concentration and separation of the minerals to obtain marketable products. The selective flotation of lead-zinc ores has been the solution to the problem in a great many cases and the results of the experimental work conducted by C. S. Parsons on this ore, show that marketable products with high recoveries can be obtained, so that one of the chief difficulties of the earlier operations has been overcome. Comparatively cheap power is also now available from the recent developments at Calumet falls, 3 miles distant. C. S. Parsons' report on the experimental work follows:—

Shipment for Experimental Purposes.—A shipment of four samples submitted by W. B. Timm, Mines Branch, Ottawa, was received at the Ore Testing Laboratories, November 15. The shipment comprised: a sample taken from the stock piles, weight 25 pounds, and representative of the ore mined; a sample of selected high-grade ore, for analysis, to determine if a marketable product could be sorted from the ore for direct shipment to the smelter; a sample of supposed ground mill feed for analysis, to obtain an idea of the assay value of the feed to the concentrator destroyed by fire; and a sample of the old concentrator tailings. From the analyses it will be seen that the latter two samples are not representative ones, especially that of the mill tailings, but they show that there were high tailing losses with the former methods of concentration.

Analyses of Samples.—The samples were designated Nos. 1, 2, 3 and 4, and gave on analyses the following values:—

Description of sample	Lead %	Zinc %	Iron %	Silver oz/ton	Gold oz/ton
No. 1—Stock pile ore.....	3.35	7.08	4.82	9.50
No. 2—Selected high-grade ore.....	16.52	28.26	8.35	39.78	0.02
No. 3—Supposed ground mill feed.....	2.12	10.04	8.93	3.15
No. 4—Mill tailings.....	1.77	12.15	8.16	8.20

Purpose of Experimental Tests.—The object in conducting experimental tests was to determine whether the ore could be concentrated by the improved methods of concentration applicable to this type of ore, with the production of marketable lead and zinc products and with high recoveries of the values. When the mine was last operated the only method available was gravity concentration on jigs and tables. The recovery of the lead and silver values was low, due to the tendency of the galena to slime during crushing operations. The iron pyrite and zinc blende being of practically the same specific gravity could not be separated by gravity means, and reported as a zinc-iron middling too low in zinc to be of market value. In order to obtain a marketable zinc product from this zinc-iron middling roasting was required to change the pyrite to the magnetic form, followed by magnetic separation. Such a method was not only costly but was hard to control, and with the exception of a few instances was not successful from an economic standpoint.

EXPERIMENTAL TESTS

The tests were conducted on sample No. 1, the stock pile ore, as oxidation had not taken place to any appreciable extent. A table test was made on the ore crushed to 20 mesh to determine the results from gravity concentration. Three flotation tests were conducted on the ore ground to varying degrees of fineness, as given in the screen tests following, to determine the results from selective flotation. The reagents used in making the flotation tests are given in a separate table, and the results of the table and flotation tests are also given in tabulated form.

Screen Tests of Flotation Tailing Showing Fineness of Grinding

Mesh	Test No. 1		Test No. 2		Test No. 3	
	Grms.	%	Grms.	%	Grms.	%
+48.....	2.5	0.5	0.1	0.0	0.3	0.1
—48+65.....	18.2	3.6	4.0	0.8	6.5	1.3
—65+100.....	77.4	15.5	19.6	3.9	39.5	7.9
—100+150.....	86.0	17.2	53.0	10.6	45.5	9.1
—150+200.....	62.5	12.7	62.0	12.4	71.5	14.3
—200.....	252.4	50.5	361.3	72.3	336.7	67.3
Totals.....	500.0	100.0	500.0	100.0	500.0	100.0

*Reagents used in Flotation Tests.—**Test No. 1—1,000 grammes ore*

Lead reagents:	5.0	pounds per ton	sodium carbonate
	0.2	"	Z cake
	0.4	"	sodium cyanide
	0.75	"	cresylic acid
Zinc reagents:	2.0	"	copper sulphate
	0.2	"	YZ mixture
	0.1	"	TT mixture
	0.01	"	pine oil

Test No. 2—1,000 grammes ore

Lead reagents:	5.0	pounds per ton	sodium carbonate
	0.1	"	QED
	0.4	"	sodium cyanide
	0.75	"	cresylic acid
Zinc reagents:	2.0	"	copper sulphate
	0.2	"	YZ mixture
	0.15	"	TT mixture
	0.01	"	pine oil

Test No. 3—1,000 grammes ore

Lead reagents:	5.0	pounds per ton	sodium carbonate
	0.2	"	Z cake
	0.25	"	sodium cyanide
	0.75	"	cresylic acid
Zinc reagents:	2.0	"	copper sulphate
	0.5	"	Neutral creosote oil No. 2. Dominion Tar & Chemical Co.
	0.01	"	potassium xanthate

SUMMARY OF RESULTS

The results from the table test show that a high-grade lead concentrate can be obtained, but with a low recovery of the lead and silver values in the concentrate. A low-grade zinc-iron middling was obtained. This test showed the difficulty of concentrating an ore of this type by gravity methods.

The flotation tests conducted were only of a preliminary nature but the separations were so readily obtained with little attention to manipulation that equally as good results, if not better, should be obtained in actual milling operations. Both the grade of the lead and zinc concentrates and the recoveries of these metals were good as well as the recovery of the silver values. One very important point has been determined, namely that fine grinding increases the proportion of silver values in the lead concentrate as can be seen by comparing the distribution of silver values in test No. 1 with that of tests Nos. 2 and 3. The screen tests show that much finer grinding was done in the latter tests. The results of fine grinding were that the silver content of the zinc concentrate was reduced from 33 ounces to 4.5 ounces per ton, thereby increasing the recovery of the silver values in the lead products from 60 to 87 per cent. This increased recovery of silver values with the lead concentrate is a very important item from an economic standpoint in the marketing of the concentrates.

CONCLUSIONS

It is reasonably safe to assume from the results of the small-scale experimental tests conducted that the ore of the Reader mine can be successfully concentrated by selective flotation with the production of high-grade lead and zinc products, and with good recoveries of the lead, zinc, and silver values.

Results of Table and Flotation Tests

Test No.	Product	Weight		Assays			Per cent of values		
		Grms.	%	Pb. %	Zn. %	Ag. oz./ton	Pb.	Zn.	Ag.
1	<i>Table test—</i>								
	Concentrate.....	50.5	2.5	63.68	6.58	139.00	48.4	2.5	33.3
	Middling.....	610.5	30.5	2.71	14.31	9.38	25.1	65.4	27.5
	Tailing.....	1,339.0	67.0	1.29	3.34	6.10	26.5	32.1	39.2
1	<i>Flotation tests—</i>								
	Lead concentrate.....	43.7	4.37	55.27	2.10	152.5	69.4	1.4	56.6
	Lead middling.....	77.2	7.72	11.20	7.63	5.5	24.8	8.3	3.6
	Zinc concentrate.....	110.7	11.07	0.50	52.21	33.1	1.6	84.3	31.1
	Zinc middling.....	53.0	5.30	0.62	4.32	6.56	0.9	3.8	3.0
	Tailing.....	715.9	71.54	0.16	0.21	0.94	3.3	2.2	5.7
2	Lead concentrate.....	95.0	9.5	34.66	4.10	91.7	89.2	5.5	79.4
	Zinc concentrate.....	162.0	16.2	1.48	38.70	8.8	6.8	88.8	13.1
	Tailing.....	748.0	74.8	0.20	0.54	1.1	4.0	5.7	7.5
3	Lead concentrate.....	59.7	6.0	46.54	2.74	134.0	80.9	2.2	72.2
	Lead middling.....	59.0	5.9	9.02	6.88	28.5	15.5	5.6	15.2
	Zinc concentrate.....	108.7	10.9	0.32	55.32	4.5	1.0	82.7	4.4
	Zinc middling.....	35.5	3.5	0.89	15.16	7.44	0.8	7.4	2.4
	Tailing.....	737.1	73.7	0.08	0.21	0.88	1.7	2.1	5.8

III

REPORTS OF INVESTIGATIONS: HYDROMETALLURGICAL LABORATORY

THE HYDROMETALLURGICAL TREATMENT OF IRON SULPHIDE ORES FOR THE PRODUCTION OF ELECTROLYTIC IRON AND THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS

R. J. Traill and W. R. McClelland

Introductory.—The preliminary experiments on this investigation, described in the report of 1923, showed results that were sufficiently promising to warrant a continuation of the investigation on a somewhat larger scale. Additional laboratory space was provided by the addition of a temporary laboratory, 40 by 18 feet, to the present chemical laboratories. This structure was ready for occupancy in March and the equipping of the laboratory with the necessary apparatus which was mostly of a special nature, occupied several months.

To assist in the carrying out of the larger tests additional technical help was required and Mr. W. R. McClelland was appointed in the capacity of junior chemist (temporary).

Many interesting problems have arisen during this investigation which cannot be covered nor incorporated in a report of this kind, and the information submitted in this report has mainly to do with the results of leaching on a larger scale under various conditions, and a general description of the apparatus used in this investigation.

Materials Used.—Because of the decided corrosive action of ferric chloride, it became necessary to make a study of various materials to find what would be the most satisfactory material in withstanding this action. Wood is quite readily decomposed by the action of hot ferric chloride. Enamelled ware withstands the action for a time, but has a tendency to crack at the temperature used and in addition the enamel is gradually worn off by the friction of the ore particles. Bitumen and asphalt coverings softened at the high temperatures. The common metals and alloys are readily dissolved by the ferric chloride. Rubber, hard or soft, is fairly satisfactory at temperatures below 50° C. but does not stand up at higher temperatures. Sulphur-coke, sulphur-asbestos and sulphur-sand mixtures withstand the action quite well but have a tendency to crack on sudden heating or cooling. Concrete impregnated with sulphur or iron oxide, seems to be the most suitable material found so far.

Two materials for construction of tanks or cells, which have proved entirely satisfactory, are acid-proof stoneware, and "ebony asbestos wood." The latter is an asbestos fibre with a binding cement, densely impregnated with some asphaltic compound. The high initial cost of the two materials would prohibit their use in a commercial-sized plant, but for the experimental work of the laboratory they were considered to be the most suitable, and have, therefore, been adopted for tanks, filters, and cells. For piping, it would appear that stoneware, bakelite, and cement-sulphur, are most satisfactory where high temperatures are used, but in the laboratory installation hard rubber has been used, as the temperature of the solutions to be transferred will be kept below 50° C. A hard rubber pump has also been used, but in commercial practice a stoneware pump would be better suited for handling the hot liquors.

Description and Plan of Layout.—The following is a brief description of the apparatus at present installed, with flow-sheet.

The leaching equipment consists of acid-proof stoneware tanks of the well-known kettle type, having a capacity of 15 gallons, and fitted with stoneware agitating-paddles which are driven by pulleys at about 80 r.p.m. The solution is heated by a special design of wire resistance heater, made in

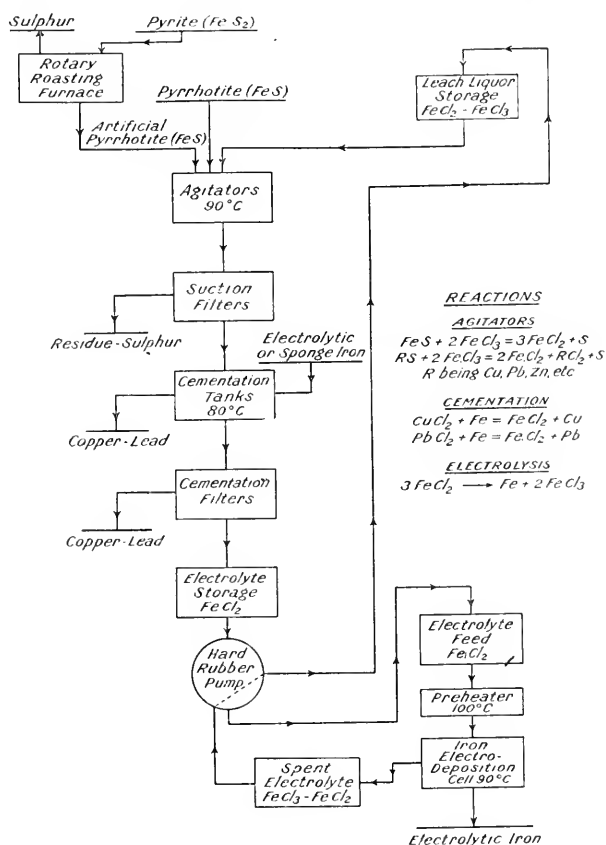


Figure 4. Flow-sheet, leaching of sulphide ores of iron and subsequent production of electrolytic iron.

the laboratory, the wire being enclosed in pyrex glass tubing. A rheostat is used to vary the current consumption from 500 to 3,000 watts. The leach is discharged from the heating tank into stoneware suction filters fitted with alundum filter-discs, one inch thick, the hot solution filtering readily with the aid of suction from a small Crowell pump. The filtered liquor is passed to the cementation tanks. These tanks are 18 inches square and made of ebony asbestos wood. In this tank the copper is precipitated, with agitation, by sponge iron (made from hematite ore and charcoal). Any lead present may also be separated in this tank. The liquor

thus purified is run through an ebony asbestos wood filter-box, 12 by 18 inches, having an alundum plate as the filtering medium. The filtered liquor is passed to the electrolyte storage tanks which have a capacity of 75 gallons, and are made of acid-proof stoneware.

When sufficient electrolyte has accumulated, it is pumped to the electrolyte feed tank, similar in capacity and design to the storage tanks. The electrolyte is now fed through a preheater to the cell. The preheater is simply a steel tube insulated and wound with a wire resistance sufficient to take about 1,500 watts. The hot electrolyte passes into the cell. The mandrel cell is built of ebony asbestos wood, with an asbestos diaphragm separating the anolyte and catholyte. The cathode is a steel mandrel 12 inches long by $3\frac{1}{2}$ inches diameter, and the anode consists of graphite plates, the side plates being about 9 by 12 inches and the bottom plate 7 by 12 inches. The speed of the mandrel can be varied from 125 to 450 r.p.m. In the catholyte compartment a hydrostatic head of about $\frac{1}{2}$ inch is maintained and the anolyte compartment is tapped at two levels for drawing off the ferric chloride formed during electrolysis. The ferric chloride thus obtained is returned by means of the hard rubber pump and piping to the leach-liquor storage tanks ready for use in leaching fresh ore.

Leaching.—The following is the general procedure carried out in the leaching process:—

The leach liquor is prepared from commercial grade ferric chloride and ferrous chloride obtained in previous leaching tests, or from the electrolytic cell, and which has been purified of copper, but not purified of zinc or calcium salts which are present in amounts too small to be separated readily. Fifty litres of leach liquor having a concentration of 64 to 74 grammes per litre ferric chloride, and a similar concentration of ferrous chloride, is prepared and run into the agitation tank, heated to about 60 to 70° C., and the ore charge, about 10 pounds, added in accordance with the particular conditions pertaining to each test. Samples of the liquor are withdrawn every 20 minutes for determination of ferrous iron, the leaching temperature being kept around 95° C. until the liquor is reduced to the ferrous condition. The reduced liquor is then discharged into the filter. The gangue and sulphur are separated, washed, dried, weighed, and analysed. Ferrous and ferric (if present) iron are determined in the filtered liquor.

Table I shows results of several leaching tests on a 50-litre scale, under various conditions; such as ore excess, concentration of ferric in leach liquor, and rate of ore feed to leach liquor. It may be observed here that the shipment of Smith pyrrhotite used in these tests contained about 10 per cent of its iron in the form of pyrite which is only slightly attacked by the ferric chloride. Analysis of the ore showed 41.6 per cent iron present as pyrrhotite, and 9.8 per cent present as pyrite, copper 0.83 per cent, and zinc 1.03 per cent, with some lime also present. The figures given in the column "per cent extraction of iron calculated from analysis of residue" are not strictly correct, as a certain amount of hydrolysis occurs during the leaching operation, and there is also a certain amount of iron precipitated by the lime in the ore, the amounts of iron so occurring in the residue being calculated as unleached iron. The actual extraction of iron, therefore, should be somewhat higher, probably by 2 to 5 per cent.

TABLE I
Leaching Tests
Ore—Smith Pyrrhotite (—200 mesh). Volume—50 litres. Temp.—93°–96° C.

Test No.	Weight of ore charge grammes	Excess of ore based on total iron present in the ore per cent	Time of agitation hours	Concentration of leach liquor in grammes of iron per litre		Rate of ore feed	Weight of residue grammes	Analysis of leached liquor in grammes of iron per litre		Extraction of copper per cent	Extraction of zinc per cent	Extraction of soluble iron calculated from analysis of residue per cent	Extraction of total iron calculated from analysis of residue per cent	Reduction of ferric iron in leach liquor per cent
				Fe''	Fe'''			Fe''	Fe'''					
X-2	4,042	25.0	14	77.3	66.4	All ore added at start.	2,048	162.5	8.4	91.0	99.5	75.2	69.2	87.4
X-3	4,400	38.3	9	58.0	65.9	All ore added at start, 13.3% excess added end of first day.	2,948	148.7	1.9	83.8	87.7	63.1	59.4	97.2
X-4	4,421	41.0	8	66.4	64.4	Ore added 10-min. intervals, 16% excess at fifth hour.	2,408	161.0	trace	86.1	94.7	75.5	67.7	100.0
X-5	4,000	25.0	7	67.9	65.9	Ore added 20-min. intervals.	2,129	160.1	1.4	88.0	93.8	79.4	69.5	97.8
X-6	4,550	25.0	7½	65.8	74.7	20-min. intervals last addition at fifth hour.	2,636	172.0	3.5	87.6	91.5	75.2	68.0	95.3
X-7	4,550	25.0	7½	68.4	74.4	20-min. intervals.....	2,636	176.6	2.5	88.0	77.7	67.8	96.6

Test X2, it will be observed, was very slow in leaching, requiring two days, and even then the reduction in solution was only 87.4 per cent. The ferric chloride used in this test was of different manufacture than that used in the tests that follow. This particular ferric chloride shipment had given similar trouble in smaller tests and the only reason that can be given in explanation is the fact that it contained ferric chloride in some colloidal form, the presence of which was discovered in making a dialysis test. The total ore charge was added at the one time. Filtration was extremely slow.

Test X3

This test was run using a new shipment of ferric chloride. The initial ore charge was 4,000 grammes, and after 5 hours' agitation the reaction slowed up to about 2 grammes of iron per hour dissolving, so an additional 400 grammes of ore was added at the eighth hour to complete the reduction of the liquor. The low iron extraction in this test is therefore accountable to this 400-gramme addition, less than 100 grammes of this addition reacting to complete reduction. The filtration in this test was faster than the previous test, but occupied several hours, in which time the reduced solution became slightly oxidized, again showing a small amount of ferric in the filtered leach. The reduction time in this test is an improvement over test X2, due, no doubt, to the new ferric chloride which was free from colloidal material. There is a pronounced slowing up of the reaction, however, after 4 or 5 hours' agitation, which may be due to the ore particles becoming coated with sulphur set free in the reaction and thereby protected from the action of the ferric chloride. To overcome this effect it was decided to add the ore at short intervals. The following tests show the results of this procedure.

Test X4

The ore was added in about 500-gramme lots every 10 minutes until 3.921 grammes (25 per cent excess) had been added. The reaction showed an increase in reduction rate, but slowed up from the fourth to the fifth hour. An additional 500 grammes was added after 6 hours' agitation, resulting in complete reduction of solution in the next two hours. In comparison with test X3, this test shows a gain in iron extraction, and the filtration rate was also improved, the time required being $1\frac{1}{2}$ hours.

Test X5

In this test the ore was added at 20-minute intervals, approximately 500 grammes at a time. The rate of reduction showed a slight increase over test X4, and although the reaction slowed up a little after 5 hours' agitation it was more progressive on the whole. The test was filtered after 7 hours' agitation, filtration being completed in 40 minutes. Figure 5 shows a curve representing the solution of iron on a time basis.

Test X6

The ore was added in 500-gramme amounts every 20 minutes until 4,000 grammes had been added, the last addition being withheld until the

slowing up period of the reaction had occurred—after about 5 hours' agitation. The fresh ore added at this stage was readily reacted on as observed by an increase in the rate of reduction of the liquor. The higher ferric concentration of leach liquor as used in this test, apparently does not materially offset the time factor in leaching. The solution of iron curve is shown in Figure 5 and compares closely with X5. The effect of the addition of fresh ore is noticeable at the 360-minute period.

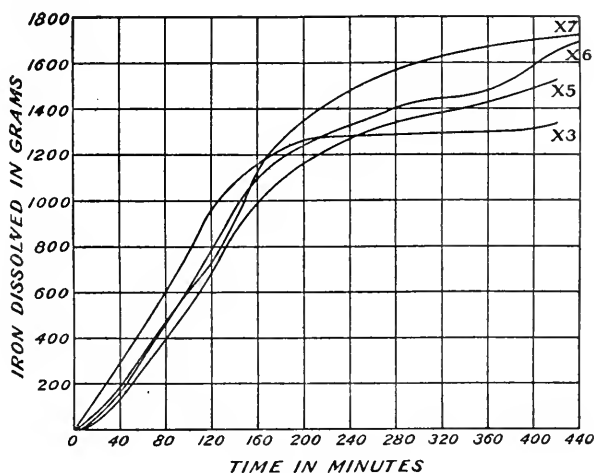


Figure 5.

Test X7

This test was run as a comparative test with test X6, using the same concentration of leach liquor (ferric) and same ore charge, but adding the charge at 20-minute intervals until all had been added. The reaction was progressive until the sixth hour; from the third hour to the sixth the test showed a higher rate of extraction than test X6. The solution of iron curve is shown in Figure 5.

CONCLUSIONS

The solution rate of iron compares favourably in all of these tests for the first three hours. The reaction tends to slow up at about this period, either because the leach liquor becomes depleted in ferric, or because the liberated sulphur adheres to the ore particles, thereby protecting them from the action of the ferric chloride. From the tests made, it would appear that the latter cause has the greater effect. The iron extractions are not so high as had been expected, but this is explained somewhat by the fact that the ore contains 10 per cent iron in the form of pyrite which scarcely reacts at all with ferric chloride.

The copper and zinc extractions are fairly satisfactory.

ROASTING OF PYRITE

In the Report of Investigations, 1923, under test No. 182, experiment showed that pyrite could be converted into an artificial pyrrhotite by roasting in a non-oxidizing atmosphere at about 750° C, one atom of recoverable sulphur being set free. A furnace was designed in an attempt to roast the ore and recover the sulphur. The furnace was of the rotary type, having a tube 4 feet long by 4 inches inside diameter, as roasting-chamber. Special bearing castings were fitted to each end of the tube with openings for feed, sulphur-gas exit, and roasted-product discharge. Provision was also made for circulation of the non-oxidizing gas through the furnace, this gas consisting mostly of sulphur dioxide (SO_2) formed in the initial stage of roasting. The furnace was heated by resistance wire elements of the muffle-furnace type, the middle section of the tube to the length of 2 feet being heated, the temperature being controlled by a rheostat. The furnace as designed was air- and gas-tight; the sulphur condenser and the roasted product receptacle being closed, the feed hopper closed by the ore, and the bearing castings in the tube furnished with gaskets. It was found that ordinary iron or steel tubes were quite unsuitable, the sulphur gas set free from the ore combining with the iron of the tube formed an iron sulphide which built up and destroyed the tube. Alundum refractory linings were tried, but, while withstanding the action of the sulphur gas, they had the tendency to crack badly. Various alloy tubes were tried, but they were all readily attacked by the sulphur gas. At present a calorized steel tube is in use and so far has proven quite satisfactory.

The drawing off and condensation of the sulphur gas is a problem that has not been satisfactorily solved, but results obtained are at least encouraging. To convert the pyrite completely to the monosulphide soluble form, it has been necessary to pass the charge through the furnace twice and occasionally three times. With a longer section of the tube heated it is quite probable that roasting would be complete in the one pass.

Test No. 182

The ore used in the roasting and subsequent leaching tests contains about 85 per cent pyrite (41.5 per cent Fe), 4.5 per cent chalcopyrite (1.6 per cent Cu), with a small amount of lead sulphide. The charge usually 30 pounds, is fed to the furnace at the rate of 15 to 20 pounds per hour, the temperature being from 750 to 800° C. After the first pass through the furnace a product analysing from 20 to 28 per cent soluble ferrous iron is obtained, which on the second pass is increased to 40 to 48 per cent soluble iron, and the third pass gives a product 48 to 52 per cent soluble iron. The sulphur recovery usually ranges from 50 to 65 per cent. Some sulphur is of course oxidized at the start of the roasting, and the ore contains ferric oxide which combines with some sulphur becoming ferrous sulphide, and sulphur also condenses in cooler parts of the furnace. The roasted product is quite grey in colour, porous in appearance, and is easily powdered by rubbing between the fingers. This last feature suggested the dispensing with further grinding before leaching, and experiment has shown that the material as obtained from the furnace is readily soluble. Lack of further grinding has, moreover, reduced to a minimum the formation of slime which had formerly been somewhat troublesome in filtration.

Leaching Roasted Pyrite.—The leach liquor was prepared from commercial ferric chloride and ferrous chloride made by treating ferric chloride with roasted pyrite and cementing out the copper and most of the lead. Leaching was carried out in 50-litre volumes, the necessary ore charge being added as outlined in the following tests. The roasted pyrite averaged 49 per cent iron, 1.8 per cent copper, and 0.7 per cent lead, with 1 to 2 per cent iron in an insoluble form, probably unaltered pyrite. Tests were made every 20 minutes to observe rate of reduction and solution of ore. Table II shows results obtained on roasted pyrite.

REMARKS

Test Y1

This test was run to obtain a supply of ferrous liquor for subsequent tests.

Test Y2

This test was run in a manner similar to the pyrrhotite tests, namely adding the ore at intervals of 20 minutes. The theoretical amount of ore required was thus added. The reduction proceeded at an even rate, but somewhat slower than with pyrrhotite. The additional excess ore was added after 5 hours' agitation, the reduction rate increasing a little thereby. It will be noticed from the curve, Figure 6, that the reaction was somewhat slow in starting, as up to the 40-minute period very little increase in ferrous iron was noted. The leach filtered in 15 minutes, being practically free from slime and therefore much easier to handle.

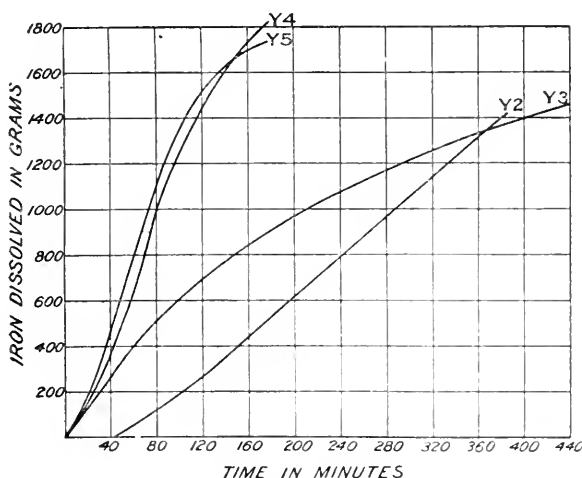


Figure 6.

Test Y3

In this test the theoretical quantity of ore required to react with the ferric chloride was used and the ore was added in total amount at the start. Reduction set in immediately and was much better than Y2 until

TABLE II

Leaching Tests

Ore—Roasted Eustis pyrite (—40 mesh). Volume—50 litres. Temp.—94°–96° C.

Test No.	Weight of ore charge grammes	Excess of ore based on Fe ⁺⁺ (soluble) present in the ore per cent	Time of agitation hours	Concentration of leach liquor in grammes of iron per litre		Rate of ore feed	Weight of residue grammes	Analysis of leached liquor in grammes of iron per litre		Extraction of copper per cent	Extraction of lead per cent	Extraction of soluble iron (Fe ⁺⁺) calculated from analysis of residue per cent	Extraction of total Fe ⁺⁺ calculated from analysis of residue per cent	Reduction of ferric in leach liquor per cent
				Fe ⁺⁺	Fe ⁺⁺⁺			Fe ⁺⁺	Fe ⁺⁺⁺					
Y-1	7,000	20	6½	0.0	116.7	20 minute intervals....	4,222	171.0	1.0	83.0	81.0	99.2
Y-2	4,846	26	11½	71.9	74.4	Theoretical at 20 min. intervals. Excess at 5th hour.	2,958	180.2	trace	71.3	100	70.2	68.3	100.0
Y-3	3,360	0	10½	69.4	64.9	All ore added at start.	1,729	160.5	2.5	95.5	100	85.2	83.7	96.2
Y-4	4,609	20	3	68.9	75.5	Two equal lots ½ hr. interval.	2,494	179.1	trace	73.9	100	74.8	100.0
Y-5	3,997	5.5	3	64.2	74.5	Two equal lots ½ hr. interval.	2,098	172.7	trace	78.2	100	77.2	100.0

the sixth hour when the rate slowed down. Had the leach liquor been of the same ferric concentration as in the former test the difference in rate of reduction would have been still more pronounced. The extraction of copper and iron is very much better. The total time is somewhat high and should be reduced by using excess ore. Filtration again proved rapid.

Test Y4

A 20 per cent excess of ore was added in this test in two equal lots at half-hour intervals. The rate of reduction was exceedingly rapid and was complete within $3\frac{1}{2}$ hours. Filtration required 25 minutes. The time of reduction is surprising and the total extraction may be considered fairly good.

Test Y5

This test was a duplicate of the above except in quantity of excess ore added, and it checks test Y4 very closely. The total extraction is slightly higher on account of less excess ore charge. Filtration required 40 minutes.

CONCLUSIONS

Tests Y4 and Y5 would indicate that properly roasted pyrite leaches more rapidly than natural pyrrhotite and does not require fine grinding. From results obtained it would appear that the addition of the total ore charge within the first half hour gives the best results. This is the opposite of the procedure found most suitable for pyrrhotite and it is suggested that this may be due to the coarser condition of the ore charge, the iron in the porous coarse particles of ore being leached out and the sulphur being left in larger masses. Microscopic examination of the dried leach residue shows a minimum of fines present, in fact the residue appears even coarser than the original ore charge. Its fineness would be difficult of determination on account of its softness and consequent disintegration on a moving screen.

PURIFICATION OF ELECTROLYTE

The filtered leach liquor contains copper and zinc in the case of the pyrrhotite, and copper and lead from the roasted pyrite. Copper is readily separated out by cementation. Sponge iron containing 88 per cent iron made in a crucible from hematite ore and charcoal was used for precipitation of the copper. Difficulty has been experienced in filtering the copper-free liquor. At present it is not known definitely whether this is due to the presence of silica in the sponge iron becoming gelatinous in the solution and thereby clogging the filter plates, or whether a too neutral stage is reached whereby some of the iron takes a colloidal form or a readily hydrolised form. An examination of the washed residue on the plate shows the presence of iron and silica. The addition of a small amount of acid generally improves filtration. A study of the hydrogen-ion concentration of the liquor at this stage of the process will, it is believed, be of some value. Lead is partly precipitated out by iron but not completely. The elimination of zinc from the electrolyte has proved somewhat difficult, and has not so far been satisfactorily solved. The leach from the pyrrhotite ore contains approximately 0.8 gramme zinc per litre. Sodium or

calcium sulphide precipitates about 25 to 40 per cent of the zinc and the limit would appear to be reached at 0.4 gramme zinc per litre. It has been suggested that the zinc be permitted to co-deposit with the iron and then allowed to volatilize in the melting process or possibly in the annealing process. The problem requires further and more definite investigation, and this will be undertaken.

ELECTRO-DEPOSITION

Several depositions of iron were made principally for the purpose of testing out the cell preliminary to the proper investigation of this phase of the process. The electrolyte used was that obtained from the leaching of the pyrrhotite and pyrite ores and contained zinc and lead as impurities, with a small amount of copper. The deposits obtained could only be considered fair, being dark grey in colour and somewhat pitted. The deposits were easily removed from the mandrel and showed silver-white on the cathode surface. The deposits contained about 2.5 to 5 per cent zinc and about 2 per cent lead, with a little copper, and were from about $\frac{1}{32}$ to $\frac{1}{16}$ of an inch in thickness.

The current density ranged from 101 to 105 amperes per square foot with a cell voltage of 4 to 4.2 volts, showing a current efficiency (corrected for impurities) of 88 to 90 per cent. The maximum temperature in the cell was 80° C., and the anolyte outflow ranged from 8 to 30 grammes ferric iron per litre. Under ideal conditions and proper regulation the anolyte outflow should have run about 64 per cent ferric iron.

IV

REPORTS ON THE WORK AND INVESTIGATIONS OF THE
CHEMICAL LABORATORY

H. C. Mabce

Chemist in Charge

The work performed in the chemical laboratory of the Division during the year was similar in classification to that outlined in reports of previous years. It consisted of regular routine analytical work performed on samples connected with the experimental treatment of ores under investigation; special chemical problems connected with the treatment of complex ores, and research work. The latter involved a continuation of investigations in the electrolytic deposition of iron begun in previous years; and the study and development of hydrometallurgical methods for sulphide ores, the details and progress in connexion with these investigations being fully dealt with elsewhere in this report by members of the staff under whose supervision the work is being conducted.

Owing to the increased number of shipments of Canadian ores received and the large amount of experimental work involved in their treatment, the necessary chemical routine work was considerably increased during the year. From January 1 to December 31, over 5,000 chemical determinations were performed on 1,511 samples of ores and concentration products. This number represents an increase of over 15 per cent of that reported in 1923.

The following list of ores and minerals indicates the variety of determinations made:—

Arsenical gold ores.....	78	Magnesite.....	5
Calcite.....	8	Molybdenite.....	135
Copper.....	183	Nickel-copper.....	8
Dolomite.....	1	Platinum-gold.....	10
Gold-silver.....	521	Silver-lead-zinc.....	17
Gold-copper.....	74	Tungsten.....	2
Graphite shale.....	6	Titanium.....	18
Lead-zinc.....	255	Zinc-lead.....	164
Magnetite (iron ore).....	11	Identification and valuation.....	19

Many of the ores, notably those from the Chu Chua district, B.C.; gold ores from The Pas district, northern Manitoba; and the gold-copper ores of northern Ontario and Quebec, presented unusual metallurgical problems, and, therefore, required an unusual amount of investigation with respect to their physical and chemical properties in order to establish suitable methods of treatment. B. P. Coyne, in addition to the regular routine analytical work, gave very valuable assistance to the engineers in this work. In July, R. A. Rogers, chemist in the Reclamation Service, Department of Interior, was transferred to the vacancy of Junior Chemist created by the resignation of D. T. Fotheringham. The addition to the present laboratory buildings, which was started late in the fall of the previous year was completed early in 1924, and installation and equipping were accomplished without any material interruption of routine work. The new addition will provide extra room for an assay laboratory and an electro-chemical research laboratory.

A COMBINED PYRO- AND HYDROMETALLURGICAL PROCESS FOR THE TREATMENT OF NICKELIFEROUS-PYRRHOTITE ORES WITH THE SUBSEQUENT RECOVERY OF IRON AND SULPHUR IN ADDITION TO NICKEL AND COPPER.

H. C. Mabee and A. E. Smaill

Introductory.—During the years that have elapsed since the process of smelting and bessemerizing as a method of treating the nickel-copper sulphide ores of Ontario was adopted, it has been fairly well demonstrated that with modern equipment and efficient management, good profits are to be obtained from the recovery of the nickel-copper values by this process. On the other hand, however, the metallurgical problems involved in the recovery and utilization of the sulphur and iron content, now entirely wasted, have never been satisfactorily solved, and anything that can be done to encourage or develop on a profitable basis a more complete recovery of the values in these ores would be an exceedingly valuable contribution to national economy and to the further development of our mineral industry in the recovery of the iron and sulphur.

Although the ores of the Sudbury district vary somewhat in their nickel-copper-iron-sulphur content, depending on the locality, the following table may be assumed to represent a fair average of their composition and to illustrate the losses by the present method of treatment.

TABLE I
Percentages and Values in One Ton of Ore

Metal	Per cent	Pounds	Price per lb.*	Value
			cents	\$ cts.
Nickel.....	3.00	60	29.35	17 61
Copper.....	1.50	30	14.42	4 33
Iron.....	40.00	800	1.10	8 80
Sulphur.....	30.00	600	1.00	6 00
Precious metals.....				1 50

*The prices assigned to nickel and copper are those quoted by Dominion Bureau of Statistics report of Mineral Production of Canada for 1923. That for iron is based on recent quotations for furnace pig, and that for sulphur is the present price of sulphur f.o.b. Gulf States.

It will be seen from the above table that, omitting the precious metal values, \$14.80, represented in the iron and sulphur content, is wasted from each ton of ore treated by present methods, in an endeavour to recover \$21.94, or the total values in nickel and copper, or, in other words, 40 per cent of the total value of the ore.

The following report contains a brief outline of the results of a series of experiments which were conducted during the year in the research laboratories of the Ore Dressing and Metallurgical Division, consisting of a combined smelting and leaching process having as the ultimate object the recovery of the sulphur, the solution and separation of practically all of the copper and a portion of the nickel, leaving a sufficient quantity of nickel in the iron residues for the production of a 2.5 to 3.5 per cent nickel-steel.

General Procedure—Smelting.—The ore was first smelted to a low-grade matte in a covered, four-burner, round gas-furnace using a No. 60 Morgan graphite crucible, and in an atmosphere as strongly reducing as possible. The flux consisted of lime or limestone, in sufficient quantity to form an easily fusible silicate slag with the gangue, and yielding a matte containing all the nickel, copper, iron, and sulphur originally present in the ore. This operation is quite the reverse of that employed at the nickel plants at Sudbury, where the smelting operations are conducted in blast furnaces with the main object of eliminating sulphur and the oxidation and ultimate slagging of the iron as iron silicate.

The tests performed at the Ore Dressing Laboratories were made on lots of 20-pound melts, and were conducted on low-grade ore from the Frood mine, No. 3, of the International Nickel Company, which had been received for the purpose of carrying on concentration tests by flotation.

The analysis of the ore, together with that of the products from the smelting operations will be found in the following table:—

TABLE II
Composition of Ore, Matte, and Slag

	Analysis per cent				
	Nickel	Copper	Iron	Sulphur wt. %	Total sulphides
Ore.....	1.35	1.45	18.79*	11.17	32.76
Matte.....	3.86	4.00	56.56	31.22	34.40
Slag.....	0.03	0.11	5.72	0.29

*This analysis of the ore does not include iron contained in the gangue material.

TABLE III
Weights and Recovery per Ton of Ore

	Nickel lbs.	Copper lbs.	Iron lbs.	Sulphur lbs.	Total sulphides lbs.
In 1 ton of ore.....	27.0	29.0	375.8	233.4	655.2
Matte recovered.....	26.6	27.5	389.0	214.6	657.7
Losses in slag.....	0.4	1.5	76.7	8.8

The slag losses, particularly in nickel and copper, may be attributed partly to very small particles of matte, and also to the low iron content of the ore. In the blast furnace practice at Sudbury, the losses amount to about 8.5 per cent of the total nickel and 9.5 per cent of the total copper. It is interesting to note in Table II that the analysis of the ore indicates a possible 32.76 per cent total sulphides and the matte shows an actual yield of 34.40 per cent. This increase as noted in Table III appears entirely in the iron, and is probably due to a part of the iron in the norite gangue combining with the sulphur and passing into the matte. The matte, particularly that with a high iron content, exhibits a very unstable character. It is readily acted upon by the oxygen of the air, especially in a moist atmosphere, the whole mass disintegrating to a fine powder, over 80 per cent of which will pass a 100-mesh screen.

The finely divided matte was then desulphurized at a temperature slightly above 450° C. until 85 to 90 per cent of the total sulphur was eliminated. Owing to the lack of proper mechanical roasting-equipment, no attempt was made to secure a complete recovery of the sulphur content. Some experimental work has been carried on in this connexion and it is hoped that when the necessary apparatus is available a satisfactory method may be worked out.

The calcined product was next subjected to a chloridizing roast at a temperature of from 450 to 500° C. Although the method of chloridization of ores and the extraction of the metal chlorides produced has been developed extensively within the last few years, the chemical reactions involved are not yet definitely known. However, the experimental data obtained from chloridizing practice has revealed valuable information regarding the influence that some solid and gaseous substances exert during the operation and it is from this information that the most favourable conditions for the successful treatment of the matte has been studied. The best results so far have been obtained by the following procedure:—

Chloridizing Roast.—The roasted product was thoroughly mixed with about 10 per cent of sodium chloride and then moistened with 5 to 10 per cent water.

The mixture was charged into a fireclay cylinder, about 5 inches in diameter and 24 inches long, with a perforated bottom through which air was forced at a low pressure. The whole was then introduced by degrees into a vertical electric furnace maintained at a temperature of from 450 to 500° C. As the temperature of the mass in the lower end of the tube rises, the moisture is gradually driven to the upper part of the mixture forming a compact cake above the roasting-zone, which acts as a blanket. As the roasting continues, however, the roasting zone progresses upwards until the salt is all decomposed when the mass becomes more spongy and porous. This operation requires about 4 to 5½ hours to complete. The mass was allowed to cool slightly, but while still quite hot was discharged into a suitable leaching receptacle and the chlorides of the metals leached out in hot water with constant stirring. Seventy-five per cent of the soluble metals passes into solution in a very short time. Leaching was continued for 2 to 3 hours and the liquid after settling was decanted, the residue washed twice with hot 5 to 10 per cent sulphuric acid solution, finally with hot water, and then sucked dry.

Results from Leaching Tests.—Table IV gives the results of leaching test from a chloridized roasted matte made from the Frood mine ore referred to in Table II. The roast was conducted on a thin bed of the material, and at a maximum temperature of 500° C.

TABLE IV
Leaching Test—Matte, Frood Mine Ore

	Pounds per ton of ore		
	Nickel	Copper	Iron
Metals in 1 ton of ore.....	27.0	29.0	378.8
Recovered in matte.....	26.6	27.9	339.0
Extracted in leaching.....	16.1	24.1	5.8
Retained in iron residue.....	9.8	2.3	379.5
Total losses in process.....	1.1	2.6	3.7

<i>Recovery</i>	Nickel	Copper	Iron
In leach solutions.....	59.40 per cent	83.17 per cent	1.5 per cent
In iron residue.....	37.00 "	8.21 "	100.0 "

It will be noticed in the results that although the extraction of copper is fairly high, considerable losses occur. It was found on a repeated test that the results obtained from chloridizing in thin beds were not so satisfactory as those from deep fairly hot masses kept under chloridizing conditions for a longer period.

In the following tables are given the analyses and results of a smelting and leaching test on a nickel-copper-pyrrhotite ore from the Shebandowan Lake nickel area, the chloridization of which was conducted in a deep mass as described above.

TABLE V
Analysis of Shebandowan Ore and Products

—	Nickel %	Copper %	Iron %	Sulphur %
Ore.....	3.82	4.02	31.7	21.5
Matte.....	6.75	7.00	50.2	35.05
Slag.....	0.06	0.10	7.57	0.51

TABLE VI
Recovery per Ton of Matte

—	Nickel lbs.	Copper lbs.	Iron lbs.	Sulphur lbs.
Contained in matte.....	135.0	140.0	1004.0	701.0
Recovered in leaching.....	101.9	138.2	11.0
Remaining in residue.....	32.4	2.8	977.9	2.78

TABLE VII
Recovery and Losses of Matte Content

—	Per cent of total			
	Nickel	Copper	Iron	Sulphur
In the leach solution.....	75.5	98.74	1.1
In the residues.....	24.0	0.28	97.4	0.39
Losses.....	0.5	0.98	1.5

These results show an almost complete extraction of the copper with comparatively small total losses, and an iron oxide product still containing approximately 3.3 per cent nickel, and a sulphur content of less than 0.40 per cent.

Recovery of the Metals in the Leach Solutions.—The leach solutions obtained from each of the small-scale tests were naturally small in quantity, and were therefore treated by regular laboratory electro-deposition methods to recover the copper and nickel content for the purpose of ascertaining and checking the leaching results, rather than the investigation of

a working method of recovery. However, some attention was given to this phase of the process, and it is expected that, as larger scale tests are undertaken and the necessary additional apparatus provided, a suitable electrolytic process will be developed under conditions approaching as nearly as possible those that would obtain in actual practice. It need scarcely be pointed out that the recovery of the platinum group metals and the additional values which they represent is also of great importance in establishing a successful process.

SUMMARY AND CONCLUSIONS

It is necessary in the preliminary smelting of the ore that the operation be conducted in an atmosphere as non-oxidizing as possible, in order to recover the sulphur and iron, and for this purpose a reverberatory or electric furnace could be operated quite successfully. By this preliminary treatment all substances such as lime, magnesia, and other gangue materials, which are not only unnecessary, but interfere ordinarily with successful chloridization, are eliminated, and the matte produced is more amenable to changes and decomposition than the original ore, or a concentration product of the ore.

It was found that the essential points governing a successful chloridizing roast were the presence of moisture, and oxygen, for the purpose of oxidizing the sulphur still present in the charge, furnishing as a result the necessary acid radicals for the liberation of chlorine from the chloridizing agent. By closely observing these conditions, an extraction of upwards of 98 per cent of the total copper and 75 per cent of the nickel was obtained without difficulty and a residue containing almost 100 per cent of the total iron content of the matte in the form of a finely divided oxide, and which still contained a sufficient amount of nickel to produce a steel, the quality of which would doubtless be much superior to that obtained from adding nickel direct to the molten steel bath.

V

CONCENTRATION OF THE LEAD-ZINC ORES OF EASTERN CANADA

C. S. Parsons

INTRODUCTORY

The steady advance in the price of lead and zinc in the metal markets of the world during the past few years has resulted in an active search for new deposits of these metals, and to the re-examination of known properties that are now idle.

There are a number of deposits of these metals in eastern Canada, the best known of which are the Tetreault mine at Notre Dame des Angles, Que.; the Reader mine on Calumet island, Que.; the Stirling property in Cape Breton, and the Federal mine in Gaspé district, Que. The author has run concentration tests on the ores from the first three mentioned properties. The ores from Notre Dame des Angles and Calumet island presented problems in concentration which were not successfully solved until the past year. Experimental work done on the ore from the Stirling mine has indicated the possibility of obtaining a high-grade zinc concentrate, but so far no satisfactory method of treatment has been developed.

CHARACTER AND TYPES OF ORES

Metallurgically the lead-zinc ores of eastern Canada may be classified into three distinct types, as follows:—

First.—Ores in which the sulphide minerals are fairly coarsely crystalline and can be freed by crushing to 40 or 60 mesh. The zinc blende has a high iron content (not marmatite) and gives a dark brown colour on grinding. The ores of Notre Dame des Angles and Calumet island, Que., may be cited as examples.

Second.—Ores in which the sulphide minerals are very finely disseminated and require extremely fine grinding. The zinc blende contains such high percentages of iron that it may be classed as marmatite. The ore of the Stirling mine, Cape Breton, is an example.

Third.—Ores in which the sulphide minerals are fairly coarsely crystalline and the zinc blende is of the light resin-coloured variety containing very little iron. The ore of the Federal Lead and Zinc Co.'s mine in Gaspé, Que., is an example.

EARLY METHODS OF CONCENTRATION

The ore at Notre Dame des Angles was discovered in 1910, and in 1912 Mr. Tetreault erected a concentrating mill in the hope of treating this complex ore by methods used in the Joplin district. The mill failed to produce a marketable grade of concentrate. In 1914 the Weedon Mining Company obtained a lease on the property and remodelled the mill. The galena was concentrated by graded crushing and tabling. A

zinc-iron middling product was obtained on tables, and oil flotation was used to save the slimed blende, but as the iron sulphide floated with the zinc, the product was too low grade to market. To eliminate the iron sulphide a magnetic separation plant was built and the non-magnetic pyrite was given a roast to convert it to the magnetic form. A considerable tonnage of zinc concentrates produced in this manner was marketed. It is said that they averaged about 42 per cent zinc. The process was costly and could not be looked upon as a commercial success. A large dump of zinc-iron middlings containing about 24 per cent zinc was left on the property and these are now being successfully concentrated by the British Metals Corporation (Canada), Ltd.

The Reader property on Calumet island was operated intermittently from 1896 to 1913. In 1910 a small concentrator was built and trial shipments were made to Newark, N.J. In 1912 a 150-ton concentrator was built, equipped with jaw crusher, coarse rolls, screens, fine rolls, jigs, Huntingdon mill, Wilfley and Overstrom tables. The mill was only operated for a few months and then closed down. The records show that a marketable lead product was made containing on the average, 65 per cent lead, 8 per cent zinc, 0.5 per cent copper, and 87 ounces silver per ton. A low-grade zinc concentrate was obtained averaging about 3.5 per cent lead, 28 per cent zinc and 18 ounces silver per ton, the remainder being iron sulphides. A sample of the mill tailings recently taken from the dump contained 1.77 per cent lead, 12.15 per cent iron, and 8.3 ounces silver, showing that the recovery of lead and silver obtained was very poor and probably did not exceed 50 per cent of either mineral.

A detailed description of the concentration tests on the zinc-lead ores of Notre Dame des Anges is given in Report No. 204, pages 8-19.

The detailed results of tests on the ores of the Reader mine, Calumet island, are given in Report No. 220, pages 87-91.

A summary report of the above tests has been published in mimeograph form, "Concentration of Lead Zinc Ores of Eastern Canada," Memorandum Series No. 21.

THE CONCENTRATION OF THE LAKE GEORGE ANTIMONY ORES

C. S. Parsons

History.—Occurrences of antimony ores in Canada are rare. The deposit at West Gore, Hants county, Nova Scotia, and the one at Lake George, York county, New Brunswick, are the only ones that have produced any tonnage of ore.

The Lake George deposits were discovered in 1863 and the property has been worked intermittently for the past 60 years by various companies. Many years ago, the property was operated by the Lake George Mining and Smelting Company and a small smelting plant was erected which produced about one ton of metal per week when in full operation. Attempts were made to concentrate the ore by hand-cobbing, and ship in lump form to Swansea, but this met with failure owing to the disseminated nature of the mineral in the vein material. Gravity concentration in jigs and vanners was tried but the slime losses were so high that this method was also abandoned. The Canadian Antimony Company in-

stalled a dry method of concentration similar to a method used extensively in France. This process consisted of heating the ore, which was then supposed to decrepitate, the flying pieces to be picked up in a current of air. No previous test work was done on the ore, and after the process had been installed it was found that the ore would not decrepitate, so the process was abandoned.

In mining the ore considerable wall rock was broken which contained arsenical pyrites. In refining operations the arsenic condensed with the antimony and produced a metal with properties objectionable to the trade. Some Montreal people took up the matter of the elimination of the arsenic. They undertook to remove the arsenic by leaching the antimony oxide produced by roasting the ore. Most of the arsenic could be removed in this way, but the method was not successful from an economic standpoint.

The mineral occurs as lenses of stibnite (antimony sulphide) in fissure veins in the slates and quartzites. The veins have been opened for a distance of a mile in length and a large number of shafts have been sunk. The deepest shaft is about 375 feet.

CONCENTRATION

The many failures recorded in the history of the operation of the antimony deposits in Canada have been partly due to want of suitable methods for the concentration of the ores. Stibnite is an extremely friable mineral and slimes very badly, and it is impossible to save these slimes by any known method of gravity concentration.

Two samples of ore from the Lake George deposits were submitted to the Ore Dressing and Metallurgical Laboratories of the Mines Branch. The first was received in 1916 from A. P. Slipp, K.C., of Fredericton, who was at that time interested in the deposits. This sample was low-grade ore from the waste dumps, and on analysis gave:—

Antimony, 3.15 per cent; arsenic, 0.28 per cent; gold and silver, trace.

The second sample was received in 1922 and was submitted by the North American Antimony Smelting Company, Ltd. This sample was claimed to be a true representative sample of the milling ore from the company's mine. The sample on analysis gave:—

Antimony, 11.65 per cent; arsenic, 0.37 per cent.

Synopsis of Concentration Test on Shipment No. 1.—A brief synopsis will be given of the work done on the concentration of the low-grade ore. Gravity concentration on jigs and tables followed by flotation of the slimes was tried, and straight flotation of the ore.

GRAVITY CONCENTRATION

There was not enough mineral freed at the coarser sizes, and the jig tailing would have had to be recrushed and re-treated. Tabling of the sized material gave the following results:—

-20+80 mesh	Concentrate.....	Antimony.....	31.10	per cent
		Arsenic.....	0.65	"
	Tailing.....	Antimony.....	0.67	"
		Arsenic.....	0.34	"
-80 mesh.....	Concentrate.....	Antimony.....	31.02	"
		Arsenic.....	2.71	"
	Tailing.....	Antimony.....	2.10	"
		Arsenic.....	0.28	"

These results show conclusively the difficulty of concentrating stibnite by gravity concentration. The slime loss is very high and there is also a corresponding loss in the coarser sizes from flat flakes of mineral that are carried into the tailing. The concentrate is also low grade and another point of importance is that the concentrate still contains a relatively high percentage of arsenic. The above tests were made on standard size jigs and tables.

FLOTATION

A number of small-scale laboratory flotation tests were made on the original sample, using a standard Callow laboratory testing unit. Various combinations of reagents were tried with the object of producing a high-grade concentrate with the elimination of the arsenic.

Test No. 1

0.6 pounds per ton hardwood creosote
0.2 " " steam distilled pine oil
10.0 " " sulphuric acid

Product	Analysis		Per cent of values	
	Antimony%	Arsenic %	Antimony	Arsenic
Concentrate.....	47.52	0.92	91.8	23.0
Tailing.....	0.24	0.24	8.2	77.0
Heads.....	2.75	0.29

Test No. 2

0.6 pounds per ton hardwood creosote
0.2 " " pine oil
3.0 " " caustic soda

Product	Analysis		Per cent of values	
	Antimony%	Arsenic %	Antimony	Arsenic
Concentrate.....	49.00	1.0	75.5	13.5
Tailing.....	0.66	0.28	24.5	86.5
Heads.....	2.58	0.31

Test No. 3

0.5 pounds per ton fuel oil from asphalt base
0.3 " " pine oil
neutral pulp

Product	Analysis		Per cent of values	
	Antimony%	Arsenic %	Antimony	Arsenic
Concentrate.....	58.52	0.24	87.6	3.5
Tailing.....	0.40	0.28	12.4	96.5
Heads.....	3.06	0.29

These tests show that even with a low-grade ore a high-grade concentrate can be obtained with a good recovery, and that it is possible to eliminate 90 per cent or more of the arsenic contained in the feed. Flotation in a neutral pulp with a fuel oil from a petroleum oil with an asphalt base gave the best results.

SYNOPSIS OF CONCENTRATION TESTS ON SHIPMENT NO. 2

These tests were run to design a flow-sheet suitable for the treatment of the ore on a basis of 100 tons daily capacity, the erection of a mill being contemplated on the results of this work.

Hand Sorting.—The ore was examined for the possibility of sorting sufficient lumps of pure stibnite to warrant hand-picking, but this was found to be impracticable.

Jigging.—Considerable stibnite was freed at $\frac{1}{4}$ -inch size and it was found possible to produce a jig concentrate containing 50 per cent antimony. A large proportion of true middling would have to be handled and the tailing would have to be recrushed. Considering the use of jigs from the standpoint of operating costs in a small mill, it was decided that their use would not be advisable owing to the necessity of recrushing machinery for the tailing and middling products, which would mean a more complicated flow-sheet.

Table Concentration.—Table concentration tests were tried on -14 to -48-mesh material. It was found possible to obtain a table concentrate by recleaning the primary table concentrate on a special cleaner deck, containing 58 per cent antimony and 0.25 per cent arsenic, but with a recovery of only 57 per cent of the antimony in the table feed. It was found after a careful examination of the table products under the microscope that tailing and middling products from sizes coarser than 24 mesh would have to be crushed again and that the middling from the finer sizes down to 48 mesh would have to be recrushed. The losses in the sizes from -35 to -48 mesh, due to flotation on the table, were very high and practically all the stibnite passing to the tailing was found to be entirely freed from the gangue.

Summary of Jigging and Tabling Tests.—It was found that by jigging or tabling carefully sized feed that had been crushed to -14 mesh, a recovery of 56 per cent of the antimony could be obtained in a concentrate averaging approximately 53 per cent antimony and 0.30 per cent arsenic. This grade of concentrate was considered too low to meet the company's requirements. By crushing to -24 mesh and tabling classified or sized feed a 58 per cent antimony concentrate could be obtained but the recovery dropped to 55 per cent and less.

Flotation Tests.—These tests were made on the original ore and consisted of a series of small-scale tests and one large-scale or tonnage test under conditions similar to actual mill operations. The ore for these tests was crushed so that 95 per cent passed 65-mesh Tyler standard screen.

SMALL-SCALE TESTS

Test No. 1.—Reagents used in this test were a mixture of coal tar and coal-tar creosote (40 and 60 per cent) and pine oil for frothing. A neutral pulp was used.

Test No. 2.—The Southwestern Engineering Company's oil KK 1, and steam-distilled pine oil for frothing. Neutral pulp used.

Test No. 3.—The Southwestern Engineering Company's oil KK 1, pine oil, and 4 pounds per ton of soda ash (pulp alkaline).

Test No. 4.—The Southwestern Engineering Company's oil KK 1, pine oil, and 4 pounds per ton of lime (pulp alkaline).

Test No. 5.—The Southwestern Engineering Company's oil KK 1, pine oil, and 10 pounds per ton of sulphuric acid (pulp acid).

Test No.	Product	Weight per cent	Analysis		Per cent of values	
			Sb %	As %	Sb	As
1	Concentrate.....	16.0	64.24	0.28	86.7
	Middling.....	2.8	6.38	1.62	1.4
	Tailing.....	81.2	1.75	0.35	11.9
2	Concentrate.....	17.7	61.80	0.62	81.4	35.6
	Middling.....	4.0	4.50	1.04	13.4	13.5
	Tailing.....	78.3	0.88	0.27	5.2	50.9
3	Concentrate.....	15.3	62.70	1.00	83.6	40.0
	Middling.....	4.1	13.10	1.44	4.7	15.4
	Tailing.....	80.6	1.65	0.21	11.7	44.5
4	Stibnite would not float in lime pulp.					
5	Concentrate.....	17.4	63.36	0.31	95.6	16.4
	Middling.....	4.3	5.94	1.29	2.2	16.7
	Tailing.....	78.3	0.35	0.28	2.2	66.9

LARGE-SCALE PILOT TESTS

A large-scale tonnage check test was made in a Callow flotation unit consisting of two rougher cells and two cleaner cells of the flat-bottom type. The ore was reduced to $\frac{1}{2}$ inch in a jaw crusher and rolls and fed to a $4\frac{1}{2}$ -foot Hardinge mill in closed circuit with a standard Dorr simplex classifier. The overflow of the classifier was -50 mesh and went direct to the flotation cells. The reagents used were a mixture of coal tar 40 per cent and coal-tar creosote 60 per cent from the Dominion Tar and Chemical Company, and sufficient pine oil to maintain a good froth. Sulphuric acid was used amounting to 5 pounds per ton of ore.

Concentrate—Antimony, 58.21 per cent; Arsenic, 0.28 per cent.

Tailing—Antimony, 0.94 per cent of total values.

Recovery—93.5 per cent of total values.

The results from the large-scale test confirmed the results obtained from the small laboratory tests. The grade of concentrate is slightly lower, but this does not mean that a concentrate of higher grade cannot be obtained. It was difficult to gauge the grade of concentrate being produced, but as the operator becomes more familiar with the operation of the flotation cells, better work and even a higher grade concentrate than 63 per cent antimony, and containing less than 0.3 per cent arsenic, may be expected.

CONCLUSIONS

The following flow-sheet was recommended as the simplest and most feasible method of concentrating the ore: "Crushing the run-of-mine ore to 2 inches in a breaker, ball milling in closed circuit with drag classifier to 65 mesh, the classifier overflow to go to a flotation unit of the pneumatic type."

In the construction of a small concentrator of 50 to 100 tons capacity, simplicity of design is important. Jigging and tabling followed by flotation would have required regrinding, classifying, sizing, and thickening units, making a complicated flow-sheet and adding greatly to the initial cost of the mill as well as to the labour, power, and other operating costs. On the other hand a mill designed for straight flotation would have a very simple flow-sheet and would occupy about one-third of the floor space. The ore from the breaker, crushed to 2 inches, could be fed directly to a ball mill operated in closed circuit with a drag classifier to produce a product approximately -65 mesh, which would go directly to the flotation units.

The grade of the final concentrate from the flow-sheet using straight flotation would be 5 per cent higher than that from the flow-sheet using jigs and tables.



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INVESTIGATIONS IN ORE DRESSING AND
METALLURGY

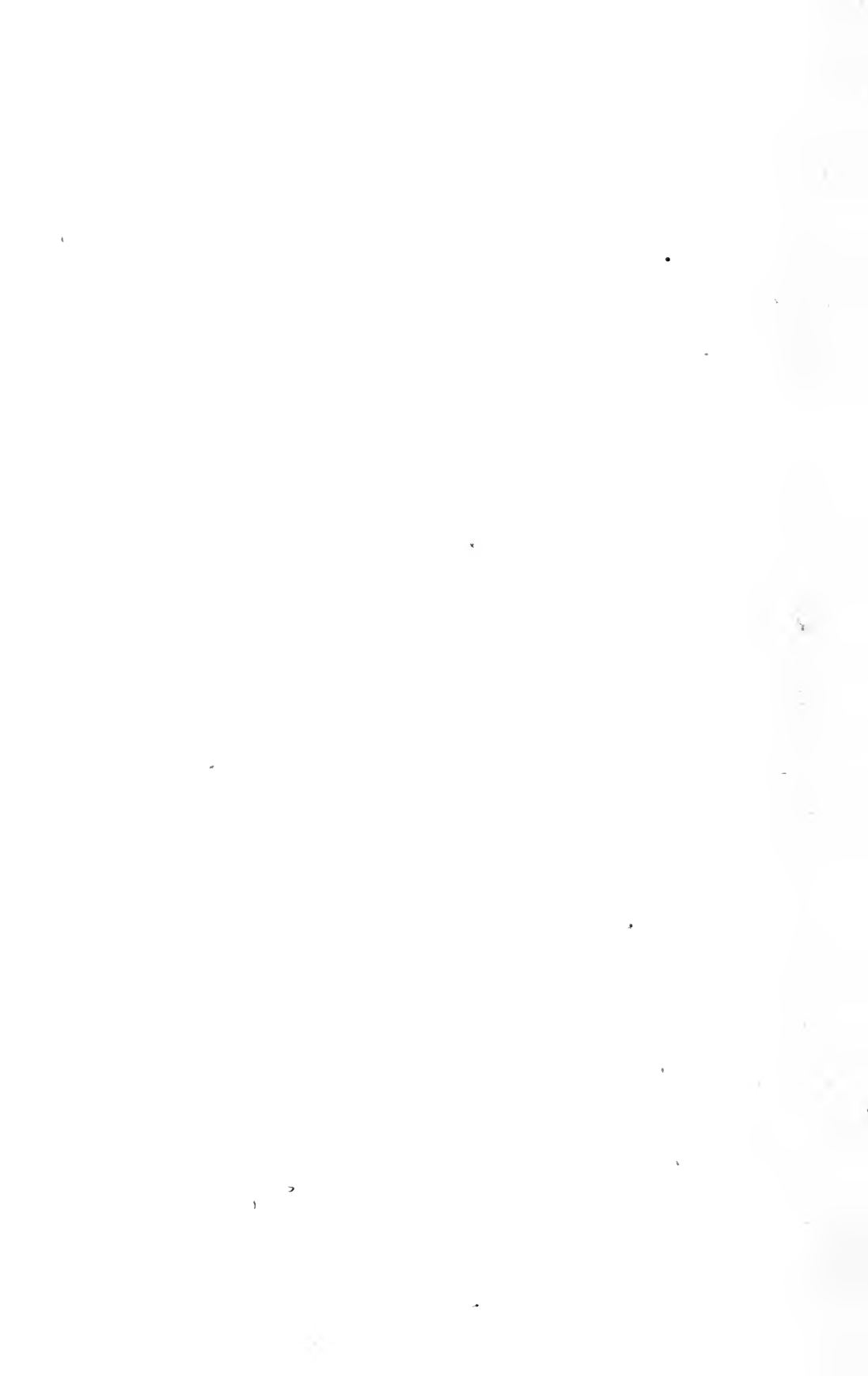
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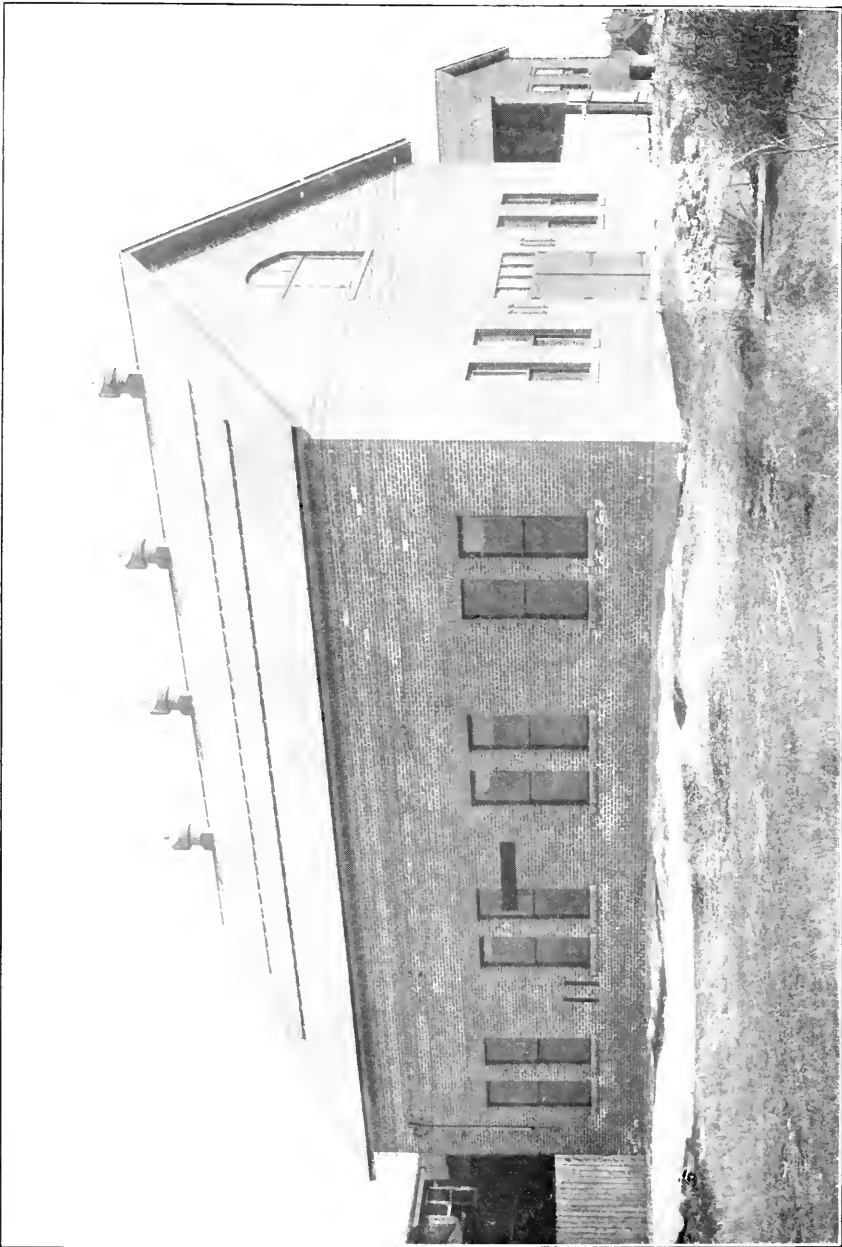
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1926





The Non-Metallic Laboratory—for investigations into the treatment and utilization of non-metallic ores and minerals. The laboratory is 60 feet by 58 feet, has two floors and is equipped with both small- and large-scale machinery and apparatus.

CANADA

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INVESTIGATIONS IN ORE DRESSING AND METALLURGY

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1926

No. 670

Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

Investigations of Mineral Resources and the Mining Industry.

Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).

Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).

Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.

1

MINES BRANCH INVESTIGATIONS IN
ORE DRESSING AND METALLURGY, 1925

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the year 1925 the laboratories of the Division were devoted entirely to experimental test and research work in connexion with the treatment of Canadian ores and metallurgical products. In order that the laboratories might be equipped with the latest apparatus for test operations new equipment was purchased, such as a Fahrenwald classifier, three new quarter-size concentrating tables, a rod mill, and a six-hearth, mechanically rabbled, roasting furnace. A new non-metallic laboratory is being equipped to study the preparation and utilization of Canadian non-metallic minerals, on completion of which more extensive experimental test work will be carried on. A pamphlet descriptive of the non-metallic laboratory will shortly be issued. On request of the National Research Council their investigatory engineers were accommodated with the use of the laboratories for their investigation of Canadian magnesites.

The investigations have been carried out under the direction of W. B. Timm, Chief of Division of Ore Dressing and Metallurgy; those in Section II on metallic ores, under the immediate supervision of C. S. Parsons, assisted by J. S. Godard, and those on non-metallic ores under R. K. Carnochan; those in Section III under the immediate supervision of R. J. Traill, assisted by W. R. McClelland; and those in Section IV under H. C. Mabee, Chief Chemist of the Division, assisted by A. E. Smail. A brief review and summary of the investigations is as follows:—

Section II (pages 11-71, inclusive)

The reports under this section include the results of the investigations conducted in the Ore Dressing Laboratories by C. S. Parsons, R. K. Carnochan, and J. S. Godard.

Report No. 221 (pages 11-13, inclusive)

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE BAIN MINE, INDIAN
LAKE, MASHAM TOWNSHIP, QUE., BY C. S. PARSONS

The ore from the Bain mine represents a type of molybdenite ore found in eastern Canada, in which the molybdenite is of the medium flake variety, associated with massive iron pyrite and pyrrhotite. It varies from the ore of the Moss mine, Quyon, Que., in which the molybdenite with an

appreciable amount of iron sulphides, is disseminated throughout a gangue of quartz, feldspar, and fluorite (altered syenite gneiss) and from the ore of the Benjamin mine, south of Amos, Que., in which the molybdenite occurs in feldspathic quartz veins with sericite. The report shows that due to the presence of so much iron sulphides, it was a more difficult ore to concentrate by flotation than the other two types mentioned. A lower grade concentrate was obtained on account of not obtaining the same differential action between the molybdenite and the gangue minerals. It was shown, however, that on an ore containing 1 per cent MoS_2 , a concentrate averaging 80 per cent MoS_2 could be obtained with a recovery of 88 per cent of the MoS_2 in the ore. Although 80 per cent MoS_2 concentrate may not be so satisfactory a product as the higher grade of 90 per cent and over for certain purposes, it is quite satisfactory for the production of ferro-molybdenum, for which purpose the greater part of the production of molybdenite concentrates is used. Included in the report is a flow-sheet followed in conducting the test. This flow-sheet will be of interest to those intending to develop this type, or any type of molybdenite deposit, as it gives a good idea of the equipment required for concentration.

Report No. 222 (pages 13-22, inclusive)

THE CONCENTRATION OF THE ZINC-IRON MIDDLING DUMP AT NOTRE DAME
DES ANGES STATION, QUE., BY C. S. PARSONS

When the zinc-iron middling dump at Notre Dame des Anges station was being concentrated by the British Metal Corporation (Canada), Ltd., a part of the dump was encountered on which poor recoveries were being made. A shipment consisting of 15 tons of the material and 15 tons of the tailings from this material was submitted to determine where the trouble lay and whether it could be overcome. The report gives the results of small-scale preliminary tests and of tonnage check tests on both samples, and shows that by using the proper reagents and control of pulp densities, etc., high-grade products could be obtained with good recoveries and that the difficulties experienced could be overcome. The report is illustrated by flow-sheets used in conducting the tests.

Report No. 223 (pages 23-28, inclusive)

THE CONCENTRATION OF SILVER-LEAD-ZINC ORE FROM THE ENTERPRISE
MINE, SLOCAN DISTRICT, B.C., BY C. S. PARSONS

The report on the concentration of this ore shows that it can be concentrated by selective flotation which gives a lead concentrate and a zinc concentrate, both marketable products, with good recoveries of the silver, lead, and zinc values in the ore. The report deals in a general way with the concentration of the Slocan ores containing lead, zinc, and silver values, the author claiming that straight selective flotation would give a simpler flow-sheet and thus lower the costs of plant installation and plant operation, and also would give higher recoveries of the values, with a greater monetary return from the sale of the products, than a combination of gravity concentration and flotation, although the latter process gives a higher grade lead concentrate containing less zinc.

Report No. 224 (pages 28-36, inclusive)

THE TREATMENT OF THE GOLD-COPPER ORE OF THE ARGONAUT MINE, LARDER LAKE, ONT., BY J. S. GODARD

The presence of copper in the gold ore of the Argonaut mine has been detrimental to good results by the cyanide process. The ground ore does not settle or filter freely, making more difficult the replacement of the dissolved values after leaching. The consumption of cyanide is high and poor recoveries are obtained. A series of tests were made to obtain a comparison of results by straight cyanidation, amalgamation, and combinations of amalgamation, concentration, and cyanidation. The tests show that the ore is amenable to amalgamation and concentration by flotation with good results and that the tailing can be cyanided with a moderate consumption of cyanide after the large bulk of the copper is eliminated by concentration. Whether cyanidation will pay for the extra recovery of the gold values after concentration is problematical. The installation of a customs smelter in the Rouyn district, Que., will assist materially the operation of this property with respect to disposal of the copper-gold concentrates.

Report No. 225 (pages 36-39, inclusive)

THE RECOVERY OF FOUNDRY METAL FROM SWEEPINGS, BY J. S. GODARD

The tests show a simple method for the recovery of metal from foundry sweepings. A simple installation of a small ball mill, screening device, and concentrating table can be used to advantage in such foundries having an accumulation of sweepings, which are being wasted or shipped out of the country for the recovery of the metal. By such a method, the metal in the sweepings can be recovered.

Report No. 226 (pages 39-42, inclusive)

AMALGAMATION AND CONCENTRATION OF A MATACHEWAN GOLD ORE, BY C. S. PARSONS AND J. S. GODARD

The report covers the treatment of the gold ore from the Young-Davidson mine at Matachewan, Ont. A large body of low-grade ore has been developed on this property. The values are too low to permit of any extensive process such as grinding to 200 mesh for cyanidation. The gold values being associated with the sulphides permits of grinding to about 30 mesh, amalgamation, and concentration of the sulphides with the recovery of the gold in the sulphides by regrinding and cyanidation. By such a process, 85 per cent of the gold values in an ore averaging \$2.80 per ton can be recovered with greatly reduced cost of treatment over an all-sliming cyanidation process. Such a process would seem applicable to other ore-bodies of a similar character.

Report No. 227 (pages 43-45, inclusive)

THE PREPARATION OF VOLCANIC ASH FROM WALDECK, SASK., FOR INDUSTRIAL PURPOSES, BY R. K. CARNOCHAN

The test work on this material included a method of grinding and separation of the ground material into various products for such uses as oil filtering, hand cleanser, household cleanser, and metal polish. A flow-sheet of a simple plant is given in the report. Such a treatment as described is now being used for the preparation of the various products given above.

Report No. 228 (pages 45-47, inclusive)

THE CONCENTRATION OF A DRY SILVER ORE FROM THE SLOCAN SILVER MINES,
LTD., ALAMO, B.C., BY C. S. PARSONS

The report gives the results obtained from a dry silver ore from the Slocan district. On No. 1 sample representing the bulk of the ore a high-grade silver concentrate can readily be obtained by flotation with recoveries of over 90 per cent of the silver values. Samples Nos. 2 and 3 being oxidized, good recoveries were not obtained. The ore represented by the latter samples was of sufficient grade to be shipped without concentration.

Report No. 229 (pages 48-50, inclusive)

THE CONCENTRATION OF A COPPER ORE FROM PITT LAKE, B.C., BY J. S.
GODARD

The report gives the results obtained on a copper ore carrying silver and gold values from Pitt Lake, B.C. The ore is very amenable to concentration by flotation grinding to 65 mesh. A high-grade copper concentrate is produced carrying the silver and gold values with recoveries between 90 and 95 per cent of the copper values.

Report No. 230 (pages 50-51, inclusive)

THE CONCENTRATION OF A COPPER-LEAD-ZINC ORE FROM ALBERT COUNTY,
N.B., BY C. S. PARSONS

The ore on which the experimental test was conducted was a mixture of fine sulphides of iron, copper, lead, and zinc, carrying some gold and silver values. The results of the test show that the ore can be concentrated into a copper-lead product carrying the most of the precious metals and over 90 per cent of the copper and lead values, and into a 50 per cent zinc product, with a recovery of over 80 per cent of the zinc values. The results were obtained by selective flotation.

Report No. 231 (pages 52-56, inclusive)

THE CONCENTRATION OF THE COPPER ORE ("C" ORE-BODY) OF THE HORNE
MINE, NORANDA MINES LTD., ROUYN, QUE., BY J. S. GODARD

The report on the concentration of the ore of "C" ore-body of the Horne mine shows that it can be concentrated with good recoveries of the copper values and fair recoveries of the gold values. In the concentration of this class of ore, selective action of the copper cannot be carried too far, as it is done at the expense of the gold values. The gold seems to be associated to some extent with the pyrite in the ore and the aim should be to make a concentrate of 12 to 15 per cent copper containing as well the pyrite in the ore, eliminating in the tailing only pyrrhotite and gangue. When such a concentrate is made it contains 95 per cent of the copper values and 80 per cent of the gold values. The economic point is one of balancing smelter costs of treating the lower grade concentrate against the loss of gold values in the higher grade concentrate and also in determining whether it is cheaper to smelt the ore direct as against concentration to this grade with the loss of 5 per cent of the copper and 20 per cent of the gold. The 20 per cent loss of gold values is less than 20 cents per ton of ore of the grade submitted.

*Report No. 232 (pages 56-58, inclusive)*THE CONCENTRATION OF A LEAD-ZINC ORE FROM THE KICKING HORSE MINE,
FIELD, B.C., BY C. S. PARSONS

The ore consisted of galena and sphalerite carrying silver values. Grinding to about 60 mesh was necessary to free the sulphides from each other and from the gangue. Selective flotation at this mesh gave good results. A recovery of 80 per cent of the lead and of 85 per cent of the zinc in a zinc concentrate assaying 50 per cent zinc could be expected. Ninety per cent of the silver values report in the two concentrates, about half in each. A higher grade lead concentrate containing less zinc could very probably be obtained in practice by recleaning the lead concentrate.

*Report No. 233 (pages 58-63, inclusive)*EXPERIMENTAL TESTS ON A GOLD ORE FROM THE CONTACT MINES, PAULSON,
B.C., BY J. S. GODARD

The experimental tests conducted comprised amalgamation, table concentration, flotation and cyanide tests, and combinations of these, to determine which would be more suitable for the treatment of the ore. Cyanidation gave good results on the sample submitted, but a change in the character of the ore, such as the presence of more copper and zinc might readily affect its treatment by this process. Over 50 per cent of the gold values is recoverable by amalgamation so that a combination of amalgamation and flotation would seem to be more applicable, especially as the property is fairly favourably situated for the disposal of concentrate to a smelter. Flotation alone gave good results, over 90 per cent of the values being recoverable in a concentrate assaying 1.6 ounces gold per ton and 2.7 ounces silver per ton, a concentration of the values of over three times that in the ore.

Report No. 234 (pages 63-64, inclusive)

THE CONCENTRATION OF A ZINC ORE FROM RENFREW, ONT., BY J. S. GODARD

The report shows that the ore is readily concentrated to a product containing over 50 per cent zinc with recoveries of about 95 per cent of the zinc values. In the samples submitted for test purposes only traces of lead were present. Other samples from the deposit showed lead to be present, in which case, selective flotation of the lead from the zinc would have to be applied in order to recover a high percentage of the lead values.

*Report No. 235 (pages 65-66, inclusive)*THE CONCENTRATION OF A LEAD ORE FROM THE FRONTENAC MINE, PERTH
ROAD, ONT., BY J. S. GODARD

The test work consisted of some small table tests to determine the grade of concentrate and recoveries that could be expected. It showed that the ore would be amenable to concentration by jigging and table concentration with good recoveries of the lead in a high-grade product.

*Report No. 236 (pages 66-68, inclusive)*THE CONCENTRATION OF A COPPER-ZINC ORE FROM THE AMULET MINE,
ROUYN, QUE., BY C. S. PARSONS

The report is interesting in that it shows that the copper-zinc ores of western Quebec can be concentrated by selective flotation into copper and zinc products. High-grade copper concentrates were obtained from this ore containing 90 per cent of the copper values. High-grade zinc concentrates were obtained indicating a possible recovery of 75 per cent of the zinc values. A great deal more research should be done on this class of ore to obtain a better selective action between the copper and the zinc with the lowering of the zinc content in the copper concentrate or in the re-treatment of the copper concentrate for the same purpose. Preliminary work along these lines should be followed by large-scale tests on a tonnage scale. The recovery of the gold and silver values is also of importance.

*Report No. 237 (pages 68-71, inclusive)*THE CONCENTRATION OF A ZINC-SILVER ORE FROM THE WONDERFUL MINE,
SANDON, B.C., BY J. S. GODARD

The ore from the Wonderful mine was being concentrated in the Alamo mill, Alamo, B.C., by gravity concentration on tables to recover the lead values, flotation of the zinc to recover the zinc values, and flotation of the slimes from gravity concentration to recover the silver values in them. This made a fairly complicated flow-sheet and the recovery of the values was poor. The results of the tests show that selective flotation without gravity concentration would give good results if the ore contained sufficient lead values to warrant such a process. The report shows that the lead content in the sample submitted is too low to produce a high-grade lead concentrate and the ore must be regarded as a zinc-silver ore. Straight flotation of the ore gives a high-grade zinc-silver concentrate containing 50 per cent zinc, 35 ounces silver per ton, with recoveries of 95 per cent of the zinc and 90 per cent of the silver values in the ore. The adoption of a simple process of straight flotation, disregarding the small amount of lead in the ore would give good results.

Section III (pages 72-88, inclusive)

The reports under this section include the results of investigations conducted in the hydrometallurgical or electrochemical laboratory by R. J. Traill and W. R. McClelland. These officers give a summary of the work accomplished during the year on "the treatment of iron sulphide ores (pyrrhotite and pyrite) for the production of electrolytic iron, with the recovery of sulphur and other metals as by-products." This report is a continuation of the work of 1924, and shows the progress made with this investigation. Progress was made on the conversion of pyrite to artificial pyrrhotite, on the purification of the electrolytes and on the electro-deposition of the iron from solution. The work on this investigation was somewhat curtailed by the taking up of a similar investigation on the treatment of Canadian ilmenites.

*Report No. 238 (pages 78-81, inclusive)***A HYDROMETALLURGICAL TREATMENT FOR PYRRHOTITE OF LOW GOLD AND COPPER CONTENT FROM THE NORANDA MINES, LTD., ROUYN TOWNSHIP, QUE., BY R. J. TRAILL AND W. R. MCCLELLAND**

The report gives a method of treatment for the massive pyrrhotite ore-bodies of the Horne mine, too low in copper and gold to be economically workable for the recovery of the copper and gold values alone, but which might possibly be worked for the recovery of the iron and sulphur content with the copper and gold as by-products. The report shows that on an ore containing 50.3 per cent iron, 1.2 per cent copper, 0.37 per cent zinc, and 0.10 ounce gold per ton, that 89.5 per cent of the iron can be recovered as pure electrolytic iron (99.96 per cent Fe), 88.6 per cent of the copper can be recovered in the process, 60.4 per cent of the sulphur can be recovered as elemental sulphur, and that the gold values have been concentrated in the residue in the ratio of 1 : 8, making this residue an ore from which the gold values and remaining copper values can be recovered in smelting operations. These results were obtained on two preliminary tests; better results as to sulphur and iron recovery could no doubt be obtained by further test and research. The report shows a possible method of treatment for this class of ore. Whether such a process would be economically possible will depend on the cost of power, plant installation, and operating costs, the latter two items determined by pilot-plant scale operations.

*Report No. 239 (pages 81-88, inclusive)***A NEW PROCESS FOR THE TREATMENT OF ILMENITE ORES FOR THE PRODUCTION OF ELECTROLYTIC IRON AND TITANIUM OXIDE CONCENTRATE FOR PIGMENT AND OTHER PURPOSES, BY R. J. TRAILL AND W. R. MCCLELLAND**

The report outlines a new process for the treatment of ilmenite ores and titaniferous iron ores, whereby the iron content in the ores is metallized and leached with ferric chloride, the iron deposited as pure electrolytic iron (99.96 per cent Fe), leaving a high-grade titanium oxide residue to be used for the manufacture of titanium pigments, titanium salts, and ferro-titanium. Results are given of the metallization of the iron to the sponge state; the leaching of the sponge; the purification of the electrolyte, and the electro-deposition of the iron. The detailed results of a test example on ilmenite ore from Ivry, Que., are given, showing that the process is applicable from a metallurgical standpoint. Sufficient laboratory research has been conducted to warrant the installation of a small pilot plant to determine cost data.

Section IV (pages 89-94, inclusive)

In this section, the progress made on the investigation into a method of treatment of heavy sulphide ores for the recovery of the iron content, as well as the other contained metals, is given. When heavy sulphide ores of copper and nickel are reduced in a furnace, under non-oxidizing conditions using an alkali flux to slag off the gangue minerals, there results a low-grade iron sulphide matte containing the copper, nickel, and precious metal values. This matte disintegrates to a fine powder in a humid atmosphere. When roasted and chloridized, the copper and a large percentage of the

nickel is rendered soluble leaving an iron oxide residue suitable for reduction to pig iron or nickel-iron pig. The soluble metals are treated for their recovery. The report contains the results obtained during the year from experimental tests on lots up to 500 pounds of ore or concentrates. When consideration is given to the rate at which iron ore reserves are being used up by industrial expansion, the time is fast approaching when the iron content in sulphide ores will have to be utilized. Canadian sulphide ore-bodies being worked for their other metal contents such as copper, nickel, zinc, and the precious metals, contain large quantities of iron, which are going out into the rock, tailing, and slag dumps. An economical process which will recover this iron in the process of treatment for the other metals would be of great national importance.

Section V (page 95)

A brief summary of the work of the chemical laboratories of the Division is given. The report shows the diversified character of the work, which was mainly on the determination of the metal values in the ores and products from test operations. B. P. Coyne, R. A. Rogers, and L. Lutes were continuously employed on this work, the first two performing the analytical and the latter the fire assay work. Considerable analytical work was also performed in the electrochemical laboratory in connexion with the electrolytic iron investigations. This included analysis of the ores and products of test operations with numerous control samples of the operations. The laboratories were also used by the National Research Council who had a technical officer working in them for a part of the year.

Section VI (pages 96-123, inclusive)

This section includes two articles of the Mines Branch Memorandum Series, Nos. 22 and 25, on Molybdenite and Graphite, and one on the ores of western Quebec. They are inserted to preserve in printed form the information contained therein. The articles on molybdenite and graphite deal with the concentration of the ores from the viewpoint of the operator and millman. The practical side of milling operations is stressed. The article on the ores of western Quebec is a continuation of Section III of the Investigations of Mineral Resources and the Mining Industry, 1924, and deals with the character of the ores and their metallurgical treatment. The articles are as follows:—

THE CONCENTRATION OF CANADIAN MOLYBDENITE ORES, BY W. B. TIMM AND C. S. PARSONS (pages 96-108, inclusive)

The report sub-divides Canadian occurrences of molybdenite ores into their several types and gives concrete examples of the results obtained from the concentration of each type of ore, the procedure followed, the flow-sheet used, grinding, control of pulp densities, reagents, and manipulation of plant machinery to obtain the best results. In a number of cases the tests were performed on carload lots of ore, under actual milling conditions, so that the results obtained are those that could be expected in practice.

THE CONCENTRATION OF CANADIAN FLAKE GRAPHITE ORES, BY C. S. PARSONS
(pages 109-118, inclusive)

The report deals with the general principles to be considered in the design of a concentrator for flake graphite ores. It gives two flow-sheets of modern plants, and a theoretical one embracing the author's opinion of the most suitable one for graphite ores in general. The author discusses the important points of each and gives details of operation of the grinding mills, flotation units, tables, filters, and dewaterers, driers, and finishing plant. The author's experience in the concentration of graphite ores, together with the experimental work conducted on a large number of graphite ores, has enabled him to stress the practical side of the operations, so that the report should be found of interest to graphite operators and those contemplating entering the field.

THE ORES OF WESTERN QUEBEC—THEIR CHARACTER AND METALLURGICAL
TREATMENT, BY W. B. TIMM
(pages 119-123, inclusive)

The report classifies the ores into three main types, gives examples of the three types on which exploration and development work has been done, and discusses the metallurgical treatment of each. The third type, comprising the large pyrrhotite ore-bodies of the district containing commercial values in gold, copper, and zinc, is of outstanding importance at present. The treatment of the ores of this latter type will require the establishment of a large smelting industry in the district.

List of Ores and Metallurgical Products on which Experimental Tests and Research Work was Conducted

In the following table is given a list of the ores and metallurgical products received at the Ore Testing Laboratories for experimental tests and research. The tabulated statement includes the report numbers, class of ore or product, source of shipment, name of shipper, and weight of shipment.

Report number	Page	Ore or product	Source of shipment	Shipper and address	Weight
					lbs.
221	11	Molybdenite....	Bain mine.....	H. H. Claudet, Rockcliffe, Ont.	61,500
222	13	Lead-zinc.....	Notre Dame mine, Que.	British Metal Corporation (Canada), Ltd., Montreal.	60,000
223	23	Silver-lead-zinc.	Enterprise mine, Silverton, B.C.	E. C. Wragge, Nelson, B.C.....	100
224	28	Gold-copper....	Argonaut mine.....	Argonaut Mines Ltd., Larder Lake, Ont.	80
225	36	Foundry sweepings.	Canada Bronze foundry, Montreal	Canada Bronze Company, Montreal.	400
226	39	Gold.....	Young-Davidson mine, Matachewan, Ont.	Porcupine Goldfields Development and Finance Co., Montreal.	16,312
227	43	Volcanic ash....	Waldeck, Sask.....	W. C. Vance, Val-Kel Cleaners, Ltd., Swift Current, Sask.	300
228	45	Silver.....	North fork Carpenter creek, Slocan district, B.C.	Slocan Silver Mines, Ltd., Alamo, B.C.	200
229	48	Copper-silver...	Pitt mine, Pitt Lake, B.C.	Pitt Mining Company, Vancouver.	218
230	50	Copper-lead-zinc.	Albert county, N.B.	John E. Teahan, Sr., Kerry P.O., N.B.	119
231	52	Copper-gold....	"C" ore-body, Horne mine.	Noranda Mines, Ltd., Rouyn, Que.	115
232	56	Lead-zinc.....	Kicking Horse mine, Field, B.C.	Pacific Mines, Ltd., Vancouver.	424
233	58	Gold.....	Contact mine, Paulson, B.C.	K.V. Mines Ltd., Vancouver....	117
234	63	Zinc.....	Pucker Street mine, Renfrew, Ont.	Alderson and MacKay, Montreal	192
235	65	Lead.....	Frontenac mine, Perth Road, Ont.	J. M. Forbes, Ottawa.....	56
236	66	Copper-zinc.....	Amulet mine, Rouyn, Que.	Amulet Mines, Ltd., Rouyn, Que.	80
237	68	Zinc-silver.....	Wonderful mine, Sandon, B.C.	Cunningham Mines, Ltd., Alamo, B.C.	235
238	78	Copper-gold....	Noranda mine, Rouyn, Que.	Noranda Mines, Ltd., Rouyn, Que.	60
239	81	Iron-titanium...	Ivry mine, Ivry, Que.	McArthur-Irwin, Ltd., Montreal	200
240	89	Copper-nickel...	Murray mine, Nickelton, Ont.	Toronto General Trust Co., Toronto.	4,000

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND METALLURGICAL LABORATORY

Report No. 221

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE BAIN MINE, INDIAN LAKE, QUEBEC

C. S. Parsons

Shipment.—A carload shipment of 30.75 tons of molybdenite ore was received March 17, 1925, from the Bain mine, at Indian lake, Masham township, Hull district, Que. The shipment was submitted by H. H. Claudet, Esq., Rockcliffe, Ottawa, Ont.

Purpose of Experimental Test.—A large-scale tonnage check test was desired to determine the amenability of this type of molybdenite ore to concentration.

Characteristics of the Ore.—The molybdenite is of the medium-large flake variety associated with a large amount of massive iron pyrite. The gangue is pyroxenite and other lime-silicate minerals. The richer parts of the ore consist almost entirely of iron sulphides. This type of ore, occurring in many parts of eastern Canada, is distinguished by the above characteristics from the more disseminated and highly siliceous types, such as the Moss mine ore, north of Quyon, Que., and that of occurrences to the south of Amos, Que. The ore showed evidence of considerable oxidization, and had apparently been taken from near the surface of the deposit.

Sampling and Analysis.—Due to the large flake and spotty nature of the ore it is very difficult to obtain an accurate sample of this type of ore. A sample of one-twentieth of the feed to the ball mill, which had been crushed to $1\frac{1}{4}$ inch by a jaw crusher, was cut by a Vezin sampler. This was reduced by graded crushing and sampling. A sample was also taken of the ground feed to the flotation cells. This wet sample showed a higher assay for the ore than the dry sample, and both samples were slightly lower than the calculated assay of the ore from the contents of the concentration products. The calculated assay of the ore showed it to contain 1.0 per cent molybdenite.

Experimental Tests

The flow-sheet, Figure 5, was followed in conducting the test. This flow-sheet has proven applicable to the concentration of most types of molybdenite ores:—

Reagents Used—

Kerosene (British American Oil Co's. "Lampolene" brand).....	0.8 pound/ton
Pine oil (steam distilled).....	0.3 " "

Screen Test of Feed to Flotation Cells—

Mesh	Weight grms.	Weight %	Cumulative %
+35.....	2.7	0.5	0.5
- 35+ 48.....	14.4	2.9	3.4
- 48+ 65.....	45.7	9.1	12.5
- 65+100.....	107.2	21.5	34.0
-100+150.....	74.0	14.8	48.8
-150+200.....	68.3	13.7	62.5
-200.....	187.7	37.5	100.0

Daily Record of Run—

Date	Description of sample	Weight of product	Analysis MoS ₂
		pounds	per cent
March 23...	Feed to flotation cells (classifier overflow).....		1.17
	Concentrates—first two hours' operation.....	26.5	76.07
	Concentrates—remainder of day.....	31.0	79.09
	Tailings.....		0.11
March 25...	Feed to flotation cells (classifier overflow).....		0.98
	Concentrates.....	122.0	81.88
	Tailings.....		0.17
March 26...	Feed to flotation cells (classifier overflow).....		1.04
	Concentrates.....	70.0	80.70
	Tailings.....		0.09
March 30...	Feed to flotation cells (classifier overflow).....		0.85
	Concentrates.....	89.5	79.84
	Tailings.....		0.11
March 31...	Feed to flotation cells (classifier overflow).....		0.74
	Concentrates.....	74.0	78.12
	Tailings.....		0.13
April 1...	Feed to flotation cells (classifier overflow).....		0.97
	Concentrates.....	66.5	81.56
	Tailings.....		0.08
April 2...	Feed to flotation cells (classifier overflow).....		0.78
	Concentrates.....	66.0	73.38
	Tailings.....		0.12
April 3...	Feed to flotation cells (classifier overflow).....		0.65
	Concentrates.....	123.0	74.46
	Tailings.....		0.18
	Clean-up of ball mill, classifier, etc.....	669.0	2.98

NOTE.—Last two days' run after ore all in ball mill and circuit being run out to obtain as much concentrate as possible; concentrates of lower grade, and tailings higher than average. First two hours' run on first day, some zinc picked up from previous operations, concentrates lower grade than average, contaminated with zinc.

Screen Analysis of Concentrates—Second Day's Run—

Mesh	Weight	Molybdenite
	per cent	per cent
+ 65.....	22.5	86.08
- 65+100.....	14.0	78.11
-100+150.....	13.0	82.80
-150.....	50.5	78.32

Analysis of Picked Flakes—

Hand-picked large flakes.....	98.50 per cent MoS ₂
Hand-picked flakes, +28 mesh concentrates.....	95.30 “

These analyses show that the flake contains a small amount of iron sulphides between the laminae, and it would be impossible to obtain a concentrate, by mechanical means, much over 90 per cent MoS₂.

Summary of Concentration Results—

One car of ore, net weight.....	61,500.0 pounds
Analysis of sample cut from dry ore by Vezin sampler.....	0.83 per cent MoS ₂
Average analysis of daily sample wet feed to cells.....	0.92 “
Analysis of feed obtained by actual weight of MoS ₂ recovered in concentrate and by calculation of tailing and clean-up.....	1.00+ “
Content in MoS ₂ , using 1.00+ per cent MoS ₂ assay.....	616.72 pounds
Amount of concentrate obtained.....	668.5 “
Analysis of total concentrate by calculation.....	78.5 per cent MoS ₂
Content MoS ₂ in concentrate.....	524.58 pounds
Clean-up from run (669 pounds at 2.98 per cent MoS ₂).....	19.94 “
Average analysis of daily tailing samples.....	0.12 per cent MoS ₂
Content MoS ₂ in tailing.....	72.20 pounds
Recovery of MoS ₂ by actual weights of products made....	87.9 per cent
Recovery figured from formula $R = \frac{100 (H-T) c}{H (C-T)}$	88.2 per cent

CONCLUSIONS

The above results show that on an ore of this type averaging 1 per cent MoS₂, a concentrate containing 80 per cent MoS₂ can be produced with a recovery of better than 88 per cent of the molybdenite values in the ore.

The ore submitted was taken from near the surface and was more or less oxidized. This state of the ore did not have any appreciable effect on the recovery, as tailings as low as 0.08 per cent MoS₂ were produced during the run. It may, however, have had some effect on the grade of the concentrate. It is possible that on fresh ore of this grade, a concentrate of 85 per cent MoS₂ could be obtained.

The production of a high-grade concentrate from this type of molybdenite ore in which the iron sulphides predominate over the siliceous gangue minerals, is more difficult than from the more highly siliceous ores. More careful control of reagents, pulp densities, deflocculation devices, etc., is required. With proper control, the results given above should be obtained, and on freshly mined ore, it would be possible to produce higher grade concentrates.

Report No. 222

THE CONCENTRATION OF THE ZINC-IRON MIDDLING DUMP AT NOTRE DAME DES ANGES, QUEBEC

C. S. Parsons

Shipment.—A carload shipment of zinc-iron middlings was received from the British Metals Corporation (Canada), Ltd. (head office, Bank of Nova Scotia Building, Montreal). The shipment contained two kinds of material, designated as Lots Nos. 1 and 2. Lot No. 1 consisted of tailings from the flotation plant which was in operation during the summer

of 1924. Lot No. 2 consisted of a part of the dump encountered just before the mill closed for the winter, and from which the tailings represented by Lot No. 1 were produced.

Location of Dump.—The dump from which these samples were taken is situated at Notre Dame des Anges, Portneuf district, Que. It was produced by the Weedon Mining Co. during the operation of the Notre Dame mine, which the company had under lease during 1915-1917. It was purchased by the British Metals Corporation in 1924, and a 100-ton flotation plant was built for the recovery of the lead and zinc. The mill was operated successfully during the summer of 1924 and produced lead concentrate and zinc concentrate, both of which were marketable.

Purpose of Experimental Tests.—Just before the concentrator was closed for the winter season, a part of the dump was encountered which had become badly oxidized. Trouble was experienced in floating this material, and in order to determine whether the results obtained could be improved upon, a sample of about 15 tons, designated Lot No. 2, representing the badly oxidized material was sent to the experimental plant of the Ore Dressing and Metallurgical Division, so that the data obtained there would be available for the resumption of operations in the spring. Lot No. 1, of about 15 tons representing the tailings obtained from this badly oxidized material, was sent to determine whether a further recovery of the zinc could be obtained by re-treatment.

Character and Analysis of Material.—Lot No. 1 contained approximately 11.5 per cent zinc as zinc blende, the balance being chiefly iron sulphides. Lot No. 2 contained approximately 24.79 per cent zinc and 2.45 per cent lead, the balance being chiefly the iron sulphides, pyrite, and pyrrhotite.

EXPERIMENTAL TESTS

Small-scale Tests on Lot No. 1

A series of small-scale tests were run before any attempt was made to operate on a tonnage scale. Different reagents and combinations were tried, the results and conclusions drawn from these tests are given below:—

Test No.	Product	Weight		Analysis Zn per cent	Distribution of Zn values per cent
		Grms.	Per cent		
1	Zinc concentrate.....	225.0	20.0	33.88	57.9
	Zinc middling.....	115.5	9.9	18.35	15.4
	Tailing.....	789.0	70.1	4.44	26.6
2	Zinc concentrate.....	165.5	16.9	51.00	68.7
	Zinc middling.....	127.0	13.9	21.64	22.4
	Tailing.....	689.4	70.2	1.59	8.9
3	Zinc concentrate.....	159.5	16.2	43.56	66.6
	Zinc middling.....	52.5	5.3	17.88	8.1
	Tailing.....	770.0	78.5	3.81	25.3
4	Zinc concentrate.....	191.0	19.2	51.40	81.6
	Zinc middling.....	60.0	6.0	16.39	8.2
	Tailing.....	745.0	74.8	1.65	10.2

Reagents Used—

Test No. 1:	10.0 lb./ton	soda ash.
	0.7	" neutral coal-tar creosote No. 2 (Dominion Tar and Chemical Co.).
	0.25	" TT mixture.
	0.05	" steam distilled pine oil.
Test No. 2:	10.0 lb./ton	soda ash.
	0.7	" neutral coal-tar creosote oil No. 2.
	0.05	" potassium xanthate.
	2.0	" copper sulphate.
	0.15	" TT mixture.
	0.05	" pine oil.
Test No. 3:	10.0 lb./ton	soda ash.
	0.7	" neutral coal-tar creosote oil No. 2.
	2.0	" copper sulphate.
	0.25	" TT mixture.
	0.05	" pine oil.
Test No. 4:	10.0 lb./ton	soda ash.
	0.05	" potassium xanthate.
	2.0	" copper sulphate.
	0.05	" pine oil.

Summary and Conclusions

The first test was run to determine whether it was necessary to add copper sulphate. It was thought that copper sulphate having been used in the previous treatment it might not be necessary to use it again. The results of tests Nos. 1 and 3 show conclusively that copper sulphate must be added. Tests Nos. 2 and 3 were run to determine the effect of the addition of potassium xanthate. The comparison of the results of these two tests suggested that potassium xanthate was producing a lower tailing. Test No. 4 was therefore made using potassium xanthate alone. The results obtained from test No. 4 were far better than any obtained in the previous experiments, and proved the value of xanthate as a reagent for the flotation of zinc.

Small-scale Tests on Lot No. 2

As on Lot No. 1, a series of small-scale tests were run, trying out different combinations of reagents. Particular attention was given to the possibility of replacing soda ash with lime, either wholly or in part, which would reduce the cost of reagents considerably owing to the large amount of alkaline necessary to neutralize the acidity of this material. The amount required varied between 15 and 18 pounds per ton. The attempt to replace the soda ash with lime was not successful. The grade of the lead concentrate was materially lowered as was also the recovery of the zinc. The results of these tests will not be given in detail with the exception of the final test No. 6, in which potassium xanthate was used. This test is given for the sake of comparison with the results of the large-scale tests:—

Test No.	Product	Weight		Analysis		Distribution of values	
		Grms	%	Pb %	Zn %	Pb %	Zn %
6	Lead concentrate.....	35.1	3.4	24.25	11.86	34.8	1.6
	Lead middling.....	56.5	5.5	10.38	22.50	24.0	4.9
	Zinc concentrate.....	365.3	35.1	1.07	53.65	15.9	76.1
	Zinc middling.....	104.5	10.2	2.20	24.13	9.4	9.8
	Tailings.....	474.5	45.8	0.82	4.17	15.9	7.6

Reagents Used—

Test No. 6: Lead—18.0 lb./ton soda ash.

	0.2	"	thiocarbamilide.
	0.25	"	sodium cyanide.
	0.20	"	cresylic acid.
Zinc—	2.0	"	copper sulphate.
	0.2	"	potassium xanthate.
	0.1	"	pine oil.

Large-scale Tonnage Check Tests on Lot No. 1 (Tailings)

The flow-sheet, Figure 1, was followed in conducting the tests on the tailings. Each test was for a period of approximately six hours. The information as to reagents used, pulp densities, samples taken, etc., and results obtained, are given in condensed form.

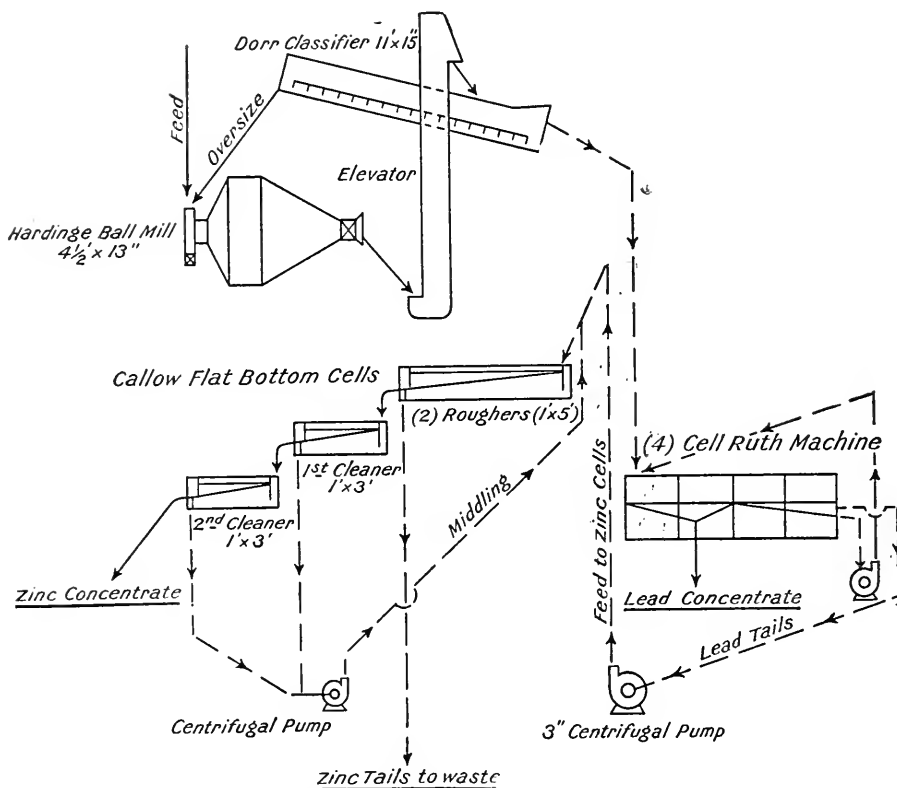


Figure 1. Flow-sheet, lot No. 1—tailings, report No. 222.

LOT NO. 1

Test No. 1

Rate of feed per hour..... 750 pounds.

Reagents—

Soda ash.....	10.00 a.m. to 12.00	8.3 lb./ton
	12.00 to 4.00 p.m.	11.0 “
Copper sulphate.....		2.5 “
Potassium xanthate.....	12.00 to 2.50 p.m.	0.07 “
	2.50 p.m. to 4.00 p.m.	0.14 “

Pulp densities—

Feed to cells, approximately 1 : 3.

Remarks—

A charge of 2,000 pounds of 1½-inch balls was used in the ball mill. The conclusion reached was that the grinding was too fine.

Key to sampling periods—

No. 1 sample.....	10.00 a.m. to 12.00
No. 2 sample.....	12.00 to 2.50 p.m.
No. 3 sample.....	2.50 p.m. to 4.00 p.m.

Analysis of samples—

Head sample, or feed.....	11.71 per cent Zn
No. 1 zinc concentrate.....	30.44 “
No. 1 zinc tailing.....	6.21 “
No. 2 zinc concentrate.....	27.59 “
No. 2 zinc tailing.....	8.59 “
No. 3 zinc concentrate.....	25.04 “
No. 3 zinc tailing.....	2.64 “

Test No. 2

Rate of feed per hour..... 865 pounds.

Reagents—

Soda ash-lime mixture (lime 3½ to soda 4)—

	10.50 a.m. to 11.30 a.m.	5.20 lb./ton
	11.30 a.m. to 12.35 p.m.	3.50 “
	12.35 p.m. to 2.10 p.m.	4.65 “
	2.10 p.m. to 2.45 p.m.	9.50 “
Soda ash.....	2.45 p.m. to 3.40 p.m.	11.00 “
	3.40 p.m. to 4.10 p.m.	Nil
Pine oil.....		0.03 “
Copper sulphate.....		2.0 “
Potassium xanthate.....		0.3 “
TT reagent.....		0.1 “

Pulp densities—

Feed to cells averaged 1 : 2.22.

Remarks.—1,500 pounds of balls was removed from the mill leaving only 500 pounds.

Key to sampling periods—

No. 1 samples.....	11.30 a.m. to 12.30 p.m.
No. 2 samples.....	12.30 p.m. to 2.20 p.m.
No. 3 samples.....	2.20 p.m. to 3.45 p.m.
No. 4 samples.....	3.45 p.m. to 4.10 p.m.

Analysis of samples—

Feed or head sample.....	10.89 per cent Zn
No. 1 zinc concentrate.....	46.62 “
No. 1 zinc tailing.....	11.45 “
No. 2 zinc concentrate.....	45.00 “
No. 2 zinc tailing.....	4.21 “
No. 3 zinc concentrate.....	40.50 “
No. 3 zinc tailing.....	4.48 “
No. 4 zinc concentrate.....	35.58 “
No. 4 zinc tailing.....	3.97 “
Analysis of total concentrate.....	42.81 “

Test No. 3

Rate of feed per hour..... 300 pounds

Reagents—

Soda ash-lime mixture (lime 2 to soda 4)—

10.20 a.m. to 12.20 p.m.	8.3 lb./ton
12.20 p.m. to 1.45 p.m.	5.0 “
1.45 p.m. to 4.45 p.m.	10.0 “

Copper sulphate..... 2.5 “

Potassium xanthate..... 10.25 a.m. to 1.45 p.m. 0.3 “

1.45 p.m. to 4.00 p.m. 0.2 “

Pine oil..... 0.03 “

Neutral cresote oil No. 2..... 4.00 p.m. to 4.45 p.m. 0.3 “

Pulp densities—

Feed to cells averaged 1: 1.25.

Key to sampling periods—

No. 1 samples..... 12.45 p.m. to 1.45 p.m.

No. 2 samples..... 1.45 p.m. to 3.25 p.m.

No. 3 samples..... 3.25 p.m. to 4.05 p.m.

No. 4 samples..... 4.10 p.m. to 4.45 p.m.

Analysis of samples—

Feed or head sample..... 11.17 per cent Zn

No. 1 zinc concentrate..... 41.03 “

No. 1 zinc tailing..... 1.60 “

No. 2 zinc concentrate..... 42.76 “

No. 2 zinc tailing..... 1.62 “

No. 3 zinc concentrate..... 38.38 “

No. 3 zinc tailing..... 1.29 “

No. 4 zinc concentrate..... 46.73 “

No. 4 zinc tailing..... 6.82 “

Total concentrates..... 41.45 “

Summary and Conclusions, Lot No. 1.—These tests show conclusively that the tailings can be re-treated and a fair grade of zinc concentrate produced with a low tailing. There seems to be no advantage in substituting lime for part of the soda ash required. Any saving in the cost of reagent is offset by a lower recovery. Potassium xanthate stands forth as a reagent of great merit; a saving in cost of reagents as compared to TT mixture is effected as well as an increase in the grade of the concentrate and in the recovery. In actual operation, judging from the results of the small-scale tests, it is safe to assume that a 50 per cent zinc concentrate can be produced, with a tailing containing less than 2.0 per cent zinc, using soda ash pulp.

Large-scale Tonnage Check Tests on Lot No. 2 (Middlings)

The flow-sheet, Figure 2, was followed in conducting the tests on the zinc-iron middlings, with the exception of test No. 6. Each test was for a period of approximately 6 hours. The information as to reagents used, pulp densities, samples taken, etc., and results obtained, is given in condensed form.

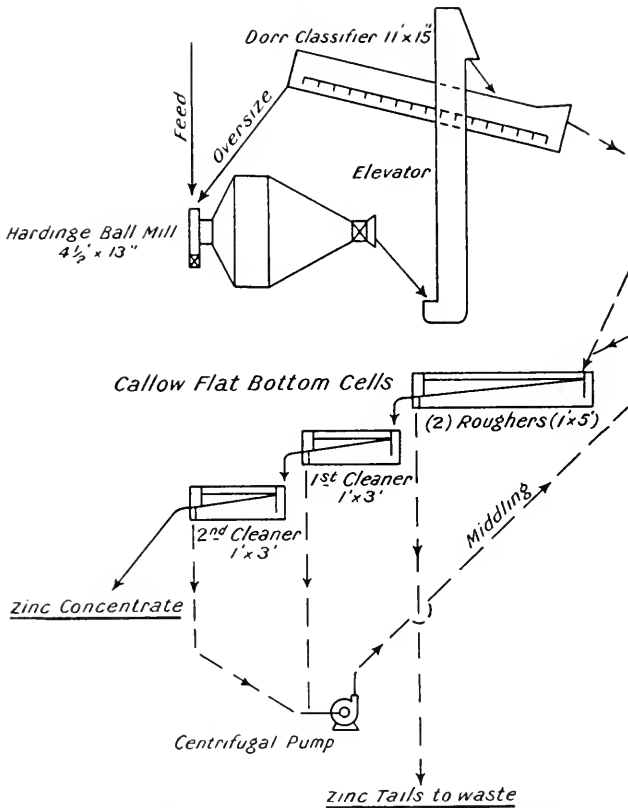


Figure 2. Flow-sheet, lot No. 2—middlings, report No. 222.

LOT NO. 2

Test No. 1

Rate of feed per hour.....	900 pounds
Reagents—	
Lead—	
Soda ash to ball mill.....	9.30 a.m. to 12.20 p.m. 18.0 lb./ton
	12.20 p.m. to 4.00 p.m. 16.0 "
Sodium cyanide.....	0.266 "
Oil mixture to ball mill (Barrett's water-gas tar, 50 per cent; D. T. & C. Co. acid creosote, 50 per cent)...	0.42 "
Zinc—	
Copper sulphate.....	2 to 2.5 "
Potassium xanthate.....	11.30 a.m. to 12.25 p.m. 0.124 "
	12.25 p.m. to 12.45 p.m. 0.37 "
	12.45 p.m. to 3.00 p.m. 0.4 "
	3.00 p.m. to 4.10 p.m. 0.12 "
Neutral creosote No. 2 (D. T. & C. Co.).....	3.00 p.m. to 4.10 p.m. 0.8 "
Pine oil.....	very little
YZ mixture—tried a little for a few minutes but did not like character of froth produced.	
Density samples—	
Average to lead cells.....	1 : 1.6
Average to zinc cells.....	1 : 2.8
Key to sampling periods—	
Lead concentrate.....	whole period
Lead tailing.....	"
Zinc sample No. 1.....	12.35 p.m. to 3.00 p.m.
Zinc sample No. 2.....	3.00 p.m. to 4.00 p.m.
NOTE.—Sample No. 2 shows the effect of replacing part of the xanthate by neutral creosote.	

Analysis of samples—		Pb%	Zn%
Feed, or head sample.....		2.45	24.79
Lead concentrate.....		16.47	12.42
Lead tailing.....		0.40	22.50
No. 1 zinc concentrates.....		0.30	51.62
No. 1 zinc tailings.....		0.45	0.85
No. 2 zinc concentrates.....		0.55	48.86
No. 2 zinc tailings.....		0.55	3.00
Zinc concentrates, whole period.....		0.55	47.13

Test No. 2

Rate of feed per hour..... 870 pounds

Reagents—

Lead—

Soda ash-lime mixture (soda 3½ to lime 4) whole run..	15.0	lb./ton
Sodium cyanide.....	0.26	"
Thiocarbamilide.....	0.20	"
Cresylic acid.....	0.40	"

Zinc—

Copper sulphate.....	2.50	"
Potassium xanthate.....	0.33	"

Pulp densities—

Feed to lead cells averaged.....	1 : 1.2
Feed to zinc cells averaged.....	1 : 3.0
Discharge to zinc cells averaged.....	1 : 15

Key to sampling periods—

Lead concentrates.....	Whole run
Lead tailings.....	"
Zinc sample No. 1.....	11.00 a.m. to 3.00 p.m.
Zinc sample No. 2.....	3.00 p.m. to 4.00 p.m.

Analysis of samples—

	Pb%	Zn%
Feed, or head sample.....	2.20	23.00
Lead concentrate.....	11.78	13.31
Lead tailing.....	0.80	24.58
No. 1 zinc concentrate.....	0.75	49.37
No. 1 zinc tailing.....	0.58	5.36
No. 2 zinc concentrate.....	1.50	52.12
No. 2 zinc tailing.....	0.67	14.40

Test No. 3

Rate of feed per hour..... 870 pounds

Reagents—

Lead—

Soda ash.....	18.0	lb./ton
Oil mixture (Barrett's water-gas tar, 50 per cent; D. T. & C. Co. acid creosote, 50 per cent).....	0.6	"
Sodium cyanide.....	0.26	"
Cresylic acid.....	0.05	"

Zinc—

Copper sulphate.....	2.5	"
Potassium xanthate.....	0.5	"

Pulp densities—

Feed to lead cells.....	1 : 1.5
Lead to zinc cells.....	1 : 3.0

Analysis of samples—

	Pb%	Zn%	Au. oz.	Ag. oz.
Feed, or head sample.....	2.35	24.60	0.05	7.16
No. 1 lead concentrate.....	22.36	9.48	0.20	49.30
No. 2 lead concentrate.....	19.46	11.82	0.15	43.95
No. 3 lead concentrate.....	20.66	10.87	0.20	46.20
Average, Cu. 4.33 per cent.....	20.81	9.39	0.25	46.54
No. 1 lead tailings.....	0.45	21.03	0.05	3.52
No. 2 lead tailings.....	0.35	24.68	0.04	3.14
No. 1 zinc concentrate.....	0.30	51.97	tr	1.20
No. 1 zinc tailings.....	0.52	7.41	0.04	3.42
No. 2 zinc concentrate.....	0.40	50.19	tr	2.20
No. 2 zinc tailings.....	0.30	3.80	0.03	2.32
No. 3 zinc concentrate.....	0.35	50.78	tr	1.82
No. 3 zinc tailings.....	0.27	0.84	0.03	2.30
Average zinc concentrate.....	0.27	52.30	tr	2.50

Test No. 4

Rate of feed per hour..... 870 pounds

Reagents—

Lead—

Soda ash, whole run..... 18.0 lb./ton
 Oil mixture (Barrett's water-gas tar, 50 per cent; D. T. & C.
 Co. acid creosote, 50 per cent)..... 0.05 “
 Sodium cyanide..... 0.18 “
 Cresylic acid..... 0.05 “

Zinc—

Copper sulphate..... 2.5 “
 Potassium xanthate..... 0.5 “

Pulp densities—

Feed to lead cells..... 1 : 1.5
 Feed to zinc cells..... 1 : 3.0

Analysis of samples—

	Pb %	Zn %	Au. oz.	Ag. oz.
Feed, or head sample.....	2.20	24.54	0.05	6.08
No. 1 lead concentrate.....	20.26	10.07	0.20	44.80
No. 2 lead concentrate.....	19.86	10.87	0.20	44.90
No. 3 lead concentrate.....	20.01	11.26	0.20	43.40
Average, Cu. 4-64.....	21.06	10.67	0.25	45.15
Lead tailing.....	0.27	26.07	tr.	1.76
No. 1 zinc concentrate.....	0.15	52.17	tr.	1.40
No. 1 zinc tailing.....	0.35	4.94	tr.	1.56
No. 2 zinc concentrate.....	0.30	50.98	tr.	0.50
No. 2 zinc tailing.....	0.47	1.53	tr.	2.48
No. 3 zinc concentrate.....	0.25	51.17	tr.	1.70
No. 3 zinc tailing.....	0.45	1.43	tr.	1.82
Average zinc concentrate.....	0.27	52.30	tr.	2.50

Test No. 5

Rate of feed per hour..... 950 pounds

Reagents—

Lead—

Soda ash..... 12.0 lb./ton
 Thiocarbamilide..... 0.2 “
 Cyanide..... 0.25 “
 Cresylic acid..... 0.05 “

Zinc—

Copper sulphate..... 2.5 “
 Potassium xanthate..... 0.5 “
 Pine oil..... 0.05 “

Pulp densities—

Feed to lead cells..... 1 : 1.5
 Feed to zinc cells..... 1 : 3.5

Key to samples—

Sample No. 1..... 11.50 a.m. to 1.45 p.m.
 Sample No. 2..... 1.45 p.m. to 2.15 p.m.
 Sample No. 3..... 2.15 p.m. to 4.20 p.m.

Analysis of samples—

	Pb %	Zn %	Au. oz.	Ag. oz.
Feed, or head sample.....	2.50	24.30		
No. 1 lead concentrate.....	18.86	11.95	0.20	42.00
No. 2 lead concentrate.....	17.96	10.27	0.15	43.45
No. 3 lead concentrate.....	17.96	10.97	0.25	40.95
Average.....	17.71	12.80		
Lead tailing.....	0.35	25.19	tr.	2.10
No. 1 zinc concentrate.....	0.45	51.37	0.01	3.10
No. 1 zinc tailing.....	0.67	1.85		
No. 2 zinc concentrate.....	0.45	51.77	tr.	2.90
No. 2 zinc tailing.....	0.47	1.70		
No. 3 zinc concentrate.....	0.20	50.88	tr.	2.46
No. 3 zinc tailing.....	0.45	0.85		
Average zinc concentrate.....	0.37	51.80		

Test No. 6

This test was run to determine whether the same results could be obtained in the Callow cells as in the Ruth cells, on the flotation of the lead.

Rate of feed per hour.....	950 pounds
Reagents—	
Lead—	
Soda ash.....	11.45 a.m. to 12.20 p.m. 14.0 lb./ton
	12.20 p.m. to 12.30 p.m. 1.4 "
	12.30 p.m. to 1.00 p.m. none
	1.00 p.m. to 3.20 p.m. 7.0 "
Thiocarbanilide.....	11.45 a.m. to 12.20 p.m. 0.20 "
	12.20 a.m. to 12.30 p.m. none
	12.30 p.m. to 1.00 p.m. 0.20 "
	1.00 p.m. to 3.20 p.m. 0.20 "
Oil mixture (Barrett's water-gas tar, 50 per cent; D. T. & C.	
Co. acid creosote, 50 per cent).....	0.30 "
Cyanide.....	0.24 "
Cresylic acid	0.05 "
Pulp densities—	
Feed to lead cells.....	1 : 2.5
Key to samples—	
Sample No. 1.....	11.45 a.m. to 12.20 p.m.
Sample No. 2.....	12.20 p.m. to 12.30 p.m.
Sample No. 3.....	12.30 p.m. to 1.00 p.m.
Sample No. 4.....	1.00 p.m. to 3.20 p.m.
Analysis of samples—	Pb % Zn %
Head sample or feed.....	2.52 25.00
Lead concentrate (whole run).....	19.06 14.70
No. 1 lead concentrate.....	16.72 15.51
No. 1 lead tailing.....	0.50 25.70
No. 2 lead concentrate.....	13.77 17.50
No. 2 lead tailing.....	0.60 25.80
No. 3 lead concentrate.....	13.27 15.40
No. 3 lead tailing.....	0.70 25.70
No. 4 lead concentrate.....	12.67 16.50
No. 4 lead tailing.....	0.65 25.30

Summary and Conclusions Lot No. 2.—The conclusions drawn from the results of these tests are summarized as follows:—

1. That there is no advantage gained in the use of lime as a substitute for a part or the whole of the soda ash. The lime not only lowered the grade of the lead concentrate, but increased the loss of zinc in the zinc tailing.

2. That higher grade lead concentrate was obtained with the oil mixture than with thiocarbanilide.

3. That by the use of xanthate for the flotation of the zinc, a very high-grade concentrate—over 50 per cent zinc—can be obtained with a very low tailing—less than 1.5 per cent zinc.

4. That there is no difficulty in obtaining a high recovery of the zinc when using xanthate under conditions of pulp densities used in these tests.

5. That the whole of the required amount of soda ash should be fed to the ball mill, thereby reducing the amount required.

6. That we find xanthate to be far superior for the flotation of the zinc as compared to TT mixture, YZ mixture, or oil mixtures of neutral coal-tar creosote or water-gas tar.

Report No. 223

THE CONCENTRATION OF SILVER-LEAD-ZINC ORE FROM THE ENTERPRISE MINE, SLOCAN DISTRICT, B.C.

C. S. Parsons

Shipment.—A shipment of 100 pounds of silver-lead-zinc ore was received from the Enterprise mine, Slocan district, B.C., on November 20, 1924. The shipment was submitted by E. C. Wragge, Esq., Nelson, B.C.

Purpose of Experimental Tests.—The shipment was submitted to determine the best method of concentration for the recovery of the silver values, and at the same time produce a lead product which would have a zinc content low enough to avoid excessive penalization by the smelter.

General Practice used on Slocan Ores.—The most serious problem in the concentration of the Slocan ores is the recovery of the silver values, in a smelting product from which the greatest monetary return will be received. The general practice throughout the district has been graded crushing with rather elaborate systems of sizing and classification of the crushed products for feed to jigs and tables on which high-grade lead products are made. The tailings and overflow slime are dewatered, the remaining fine lead and silver values in the slime partly recovered by flotation in lead cells, the tailings from which go to zinc cells for the recovery of the zinc by flotation. By this method the silver losses are appreciable, due to reasons which are given later in this report. In some cases where this general procedure is followed, but on a less elaborate scale, the silver losses amount to 60 per cent of the silver values in the ores.

Character and Analysis of Sample.—The sample received was very high-grade. The analysis shows it to contain:—

Lead.....	9.92 per cent
Zinc.....	34.42 “
Silver.....	24.91 ounces per ton
Gold.....	0.04 “

The high zinc content makes it extremely difficult to produce a lead concentrate low in zinc by flotation. The sulphides are coarsely crystalline and separate readily from each other and the gangue by crushing to 60 mesh. The gangue is chiefly carbonate of iron (spathic iron) and siliceous material.

Examination of Ore as to Silver Values.—A careful study was first made to determine the association of the silver. Pure pieces of galena, blende, and gangue, approximately $\frac{3}{8}$ inch in size, were picked out and assayed for silver. The results were as follow:—

Galena crystallized in large cubes.....	100.7 ounces per ton
Galena finely disseminated aggregates of crystals.....	102.0 “
Zinc blende (0.16 per cent lead, 63.30 per cent zinc).....	7.5 “
Gangue (trace 1.66 per cent).....	1.18 “

The assays show conclusively that the silver in some form is associated almost exclusively with the galena. Samples of the ore were then crushed to about 20 mesh and a lead concentrate made by tabling. This concentrate contained 70 per cent lead but only 67 ounces silver per ton as against 100 ounces per ton in the coarse unbroken pieces of pure galena. Other tests showed that the finer the galena was crushed the less

silver it contained. It is obvious that the silver was freed by crushing. This observation was confirmed by the results of the flotation tests, where the largest proportion of the silver was generally found in the zinc concentrate. The silver which is freed from the galena by crushing is evidently in an extremely fine form. This opinion is further borne out by some work on the Silversmith ores, which proved that 75 per cent of the silver in the zinc flotation concentrates was in the -200-mesh material, and that the silver losses in the tailing were due to very finely divided silver mineral; therefore, owing to this fine condition of the silver, a high recovery would not be expected with the galena concentrate on tables. This is proven by practice. Large losses would also occur in thickening the table tailing for flotation of the zinc, due to the tendency of the fine particles of silver mineral to remain in suspension in the overflow water.

It has been shown by experience that a mineral in such a finely divided state as this silver is believed to be in, is very difficult to recover. This applies to flotation as well as to gravity concentration. The writer has had occasion in a number of cases to observe that very fine particles of mineral in a flotation pulp repel the oil charges, and even if oiled, do not exert enough force to enter the interfaces of the air bubbles of the froth, in other words, they are difficult to oil and collect in the froth. There is, however, a marked tendency for these very fine grains to form aggregates around the large ones to which they seem to cling tenaciously. Experience with other ores has led to the opinion that to collect and recover a mineral which is in this state, a large proportion of large grains must be present in the pulp, so that by the bulk action of the flotation of the coarse grains the fine grains are gathered in and collected.

For these reasons it was decided to confine most of the experimental work to straight selective flotation with the production of a lead and zinc concentrate, and to endeavour by the bulk action, to make a high recovery of the silver, and at the same time produce a lead concentrate sufficiently free from zinc to avoid being penalized to such an extent by the smelter that the economic gain from the increased recovery of the silver would not be offset by penalization charges for the zinc content.

Experimental Tests

A number of selective flotation tests were made, first on the sample as submitted, and second, on the ore diluted by the addition of some barren quartz. It was thought that the sample submitted was higher in grade than would be the case with the milling ore, so that 50 per cent by weight of barren quartz porphyry was added to make up the sample used in the latter tests. It was believed that the lead concentrate produced would contain much less zinc than that obtained from the first series of tests on the high-grade sample submitted. A comparison of the results shows this to be the case. The results from the two series of tests are given in the following table:—

Results of First Series of Selective Flotation Tests on Sample as Submitted

Test No.	Product	Weight		Analysis				Per cent of values				Reagents used, etc.
		Grms.	%	Pb %	Zn %	Ag. oz./ton	Pb	Zn	Ag			
1	Lead concentrate.....	91.5	9.0	63.43	15.44	109.7	59.8	4.5	37.8	Lead: added to ball mill, 7 lb./ton soda ash, 0.25 lb./ton water-gas tar and coal-tar creosote; added to cells, 0.5 lb./ton cyanide, 0.05 lb./ton cresylic acid. Zinc: 2 lb./ton copper sulphate, 0.6 lb./ton neutral creosote, 0.2 lb./ton TT mixture.		
	Lead middling.....	132.5	13.1	25.05	40.44	50.9	34.1	16.9	25.5			
	Zinc concentrate.....	324.5	31.9	0.90	61.58	18.2	3.0	63.1	22.8			
	Zinc middling.....	141.5	13.9	1.00	22.23	16.2	1.4	9.9	8.6			
	Tailing.....	325.7	32.1	0.50	5.37	4.3	1.7	5.5	5.3			
2	Lead concentrate.....	82.0	8.2	70.54	9.90	89.1	63.0	3.1	28.8	Lead: added to ball mill, 3.5 lb./ton soda ash, 0.2 lb./ton thiocarbamide; added to cells 0.05 lb./ton cresylic acid, 0.6 lb./ton cyanide. Zinc: 1 lb./ton copper sulphate, 0.1 lb./ton potassium xanthate.		
	Lead middling.....	48.5	4.9	51.05	19.50	38.7	26.9	3.6	7.5			
	Zinc concentrate.....	272.0	27.3	1.55	62.07	34.7	4.6	64.9	36.8			
	Zinc middling.....	40.4	4.1	3.01	21.98	36.8	1.3	3.4	5.9			
	Tailing.....	553.5	55.5	0.70	11.71	9.6	4.2	24.9	21.0			
3	Lead concentrate.....	110.0	11.7	52.15	24.20	77.80	69.0	8.4	36.0	Lead: added to ball mill, 8 lb./ton soda ash, 0.15 lb./ton coal-tar creosote; 1 lb./ton water-gas tar; added to cells, 1 lb./ton cyanide; 0.05 lb./ton cresylic acid. Zinc: 0.6 lb./ton neutral creosote; 0.2 lb./ton TT mixture, 0.05 lb./ton No. 5 pine oil.		
	Lead middling.....	50.0	5.3	28.26	28.17	42.90	16.8	4.4	9.0			
	Zinc concentrate.....	315.0	33.3	2.45	59.72	31.3	9.2	59.0	41.0			
	Zinc middling.....	86.5	9.1	2.15	50.14	17.0	2.2	13.6	6.0			
	Tailing.....	384.5	40.6	0.60	12.10	5.0	2.8	14.6	8.0			
6	Lead concentrate.....	132.5	13.3	36.87	23.21	117.1	63.5	8.9	58.5	Lead: 5 lb./ton soda ash; 0.2 lb./ton thiocarbamide, 0.4 lb./ton special reagent, 0.05 lb./ton cresylic acid. Zinc: 0.6 lb./ton No. 2 neutral creosote, 1 lb./ton copper sulphate, 0.05 lb./ton pine oil No. 5.		
	Lead middling.....	113.0	11.3	21.74	40.17	35.5	31.8	13.3	15.1			
	Zinc concentrate.....	355.2	38.5	0.55	62.09	14.1	2.7	69.8	20.4			
	Zinc middling.....	49.1	4.9	0.90	44.84	13.8	0.6	6.4	2.5			
	Tailing.....	318.7	31.9	0.33	1.77	2.9	1.4	1.6	3.5			
7	Lead concentrate.....	130.0	12.9	61.43	16.60	120.5	78.2	6.3	57.8	Lead: added to ball mill, 6 lb./ton soda ash, 0.2 lb./ton thiocarbamide; added to cells, 0.2 lb./ton special reagent, 0.05 lb./ton cresylic acid. Zinc: 0.4 lb./ton potassium xanthate, 1 lb./ton copper sulphate, 0.05 lb./ton pine oil.		
	Lead middling.....	65.0	6.4	18.71	32.53	29.8	11.9	6.2	7.2			
	Zinc concentrate.....	343.5	34.1	1.15	60.56	15.0	3.9	61.1	19.0			
	Zinc middling.....	180.0	17.8	2.99	47.62	18.3	5.3	25.1	12.2			
	Tailing.....	290.0	28.8	0.20	1.57	3.56	0.6	1.3	3.8			
8	Lead concentrate.....	105.0	10.5	56.09	20.50	129.0	62.8	6.8	51.3	Lead: added to ball mill, 6 lb./ton soda ash, 0.2 lb./ton thiocarbamide; added to cells, 0.05 lb./ton cresylic acid. Zinc: 0.3 lb./ton potassium xanthate, 2 lb./ton copper sulphate, 0.05 lb./ton cresylic acid.		
	Lead middling.....	126.0	12.6	23.45	18.15	38.8	32.9	7.3	18.5			
	Zinc concentrate.....	469.0	47.0	0.72	56.80	14.1	3.6	84.8	25.1			
	Zinc middling.....	297.0	29.8	0.22	1.15	4.54	0.7	1.1	5.1			
	Tailing.....											
10	Lead concentrate.....	No flotation of lead under these conditions.....										

Lead: added to mill, 6 lb./ton lime, 0.3 lb./ton thiocarbamide; added to cells, 2 lb./ton sodium sulphite, 0.05 lb./ton cresylic acid.

Results of Second Series of Selective Flotation Tests on Ore Diluted with 50 per cent Barren Quartz

Test No.	Product	Weight		Analysis			Per cent of values			Reagents used, etc.
		Grms.	%	Pb %	Zn %	Ag. oz./ton	Pb	Zn	Ag	
12	Lead concentrate.....	53.0	5.3	61.85	13.85	69.30	75.4	4.4	31.2	Lead: added to ball mill, 6 lb./ton soda ash, 0.4 lb./ton thiocarbamide; added to cells, 0.3 lb./ton cyanide, 0.01 lb./ton cresylic acid. Zinc: 2 lb./ton copper sulphate, 0.4 lb./ton potassium xanthate, 0.05 lb./ton pine oil.
	Lead middling.....	30.0	3.0	24.94	25.65	46.80	17.2	4.5	12.0	
	Zinc concentrate.....	274.0	27.5	0.85	52.00	20.50	5.4	84.5	47.8	
	Tailing.....	639.5	64.2	0.13	1.75	1.66	1.9	6.6	9.0	
13	Lead concentrate.....	40.4	4.0	70.07	5.90	69.90	56.9	1.4	20.9	Lead: added to ball mill, 6 lb./ton soda ash, 0.2 lb./ton thiocarbamide; added to cells, 0.3 lb./ton cyanide, 0.01 lb./ton cresylic acid. Zinc: 2 lb./ton copper sulphate, 0.5 lb./ton potassium xanthate, 0.05 lb./ton pine oil.
	Lead middling.....	35.7	3.5	43.96	19.00	47.20	31.5	3.9	12.5	
	Zinc concentrate.....	254.0	25.0	1.35	59.80	29.70	8.4	86.4	55.8	
	Zinc middling.....	43.7	4.3	1.75	16.50	9.60	1.5	4.1	3.1	
	Tailing.....	642.0	63.2	0.13	1.15	1.64	1.7	4.2	7.8	

SUMMARY

It will be observed that in each of these tests there is a lead middling product which carries a large portion of the lead values. We have found that in continuous operation on a tonnage scale the middling takes care of itself by being returned continuously to the feed end of the cells. The following examples are cited to support this:—

Results of Small- and Large-scale Tests on a Lead-Zinc Ore:—

Product	Weight	Analysis		Per cent of values	
		Pb %	Zn %	Pb.	Zn.
Small-scale test—					
Lead concentrate*.....	35.1	24.25	11.86	34.8	1.6
Lead middling.....	56.5	10.38	22.50	24.0	4.9
Zinc concentrate.....	365.3	1.07	53.65	16.0	76.0
Zinc middling.....	104.5	2.20	24.13	9.4	9.8
Zinc tailing.....	474.4	0.82	4.17	15.8	7.7
Large-scale test—					
Lead concentrate*.....		22.36	9.39	Approximate recoveries: lead 85 per cent, zinc 90 per cent.	
Zinc concentrate.....		0.27	52.30		
Tailing.....		0.30	1.50		

*It may seem that the lead concentrates in the above tests are low-grade, but in this particular case they carry high values in silver, one-half ounce in gold, and some copper values. The ore contained considerable micaceous material. It was, therefore, important that the precious metal values together with the copper and the floatable micaceous material report with the lead concentrate, and thus permit of a much higher grade zinc product being obtained. The lead product, while low in lead values, was a very desirable smelter product.

Notes on Reagents Used.—Thiocarbanilide is apparently a satisfactory reagent for the flotation of the lead, but care should be taken not to use over 0.2 pound per ton as any excess will raise zinc. The silver content of the lead concentrate seems to be governed by the kind and quantity of reagents used to obtain the selective action between the lead and zinc. Four different modifying reagents were experimented with, namely, cyanide, sodium sulphite, sodium di-phosphite, and a special reagent. Cyanide and the special reagent were the only ones which produced any decided selective action between the lead and zinc, but the special reagent produces a lead concentrate containing more silver, thereby increasing the recovery of the silver in the lead concentrate instead of in the zinc concentrate. Soda ash is necessary, as well as copper sulphate. Xanthate is undoubtedly one of the best reagents for the zinc. We recommend the use of a thick pulp of 1 : 2.5 for the lead, and 1 : 3 for the zinc.

CONCLUSIONS

In suggesting that straight selective flotation would seem to have great possibilities on this ore, the writer realizes that he is unfamiliar with local conditions. The economic side will have to be studied by the operators who know these conditions best. He believes that a calculation of the smelter rates will show that the increased recovery of the silver, which is 92 per cent in the case of the second series of tests, will offset the penalty charges for 15-20 per cent zinc in the lead concentrate, and show a considerable margin. It must be remembered that these results have been

obtained by small laboratory tests, but we believe that there is enough merit in them to warrant serious consideration being given to the possibility of treating the ore by selective flotation.

It is suggested, in order to settle the question definitely as to which is the better method of treatment for Slocan ores, i.e., straight selective flotation as against gravity concentration and flotation, that a small pilot plant be built by co-operative arrangement among the mine owners, at one of the large mills of the district, using the latter process. This small unit of 12 to 15 tons daily capacity should be arranged so that it would receive a representative tonnage sample cut from the feed to the main mill, in order that a definite comparison of the two methods could be obtained.

Report No. 224

THE TREATMENT OF THE GOLD-COPPER ORE OF THE ARGONAUT MINE, LARDER LAKE, ONT.

J. S. Godard

Shipments.—A shipment of gold-copper ore, weighing 80 pounds, was received December 15, 1924, from the Argonaut Mine, Larder Lake, Ont.

Purpose of Experimental Tests.—It was desired to compare the merits of various methods of treatment for the extraction of the gold values contained in the ore.

Sampling and Analysis.—The entire lot was crushed to $\frac{1}{2}$ -inch, cut in a riffle sampler to 20 pounds, which was crushed to -8 mesh and cut to 4 pounds. This quantity was ground dry in a pebble mill to -150 mesh and then cut to obtain an assay sample:—

Gold.....	1.06 ounce per ton
Copper.....	2.21 per cent
Iron.....	11.23 "

EXPERIMENTAL TESTS

Cyanidation

Four single-cycle, cyanidation tests of the bottle type were made. The ore was ground dry in a pebble mill and screened through 150 mesh. The cyanide solution was strengthened to 0.05 per cent cyanide at 8-hour intervals. Following are the results of the tests:—

Test No.	Ore grms.	Solution c.c.	KCN per cent	CaO lb./ton	Assay		Extraction, per cent	Consumption lb./ton		Time agitated, hours
					Heads	Tails		KCN	CaO	
1	200	1000	0.05	3	1.06	0.08	92.4	8.0	1.8	24
2	200	1000	0.05	3	1.06	0.04	96.2	13.0	1.8	48
3	200	600	0.05	3	1.06	0.24	77.4	6.9	2.0	24
4	200	600	0.05	3	1.06	0.05	95.2	11.1	2.1	48

The tailings were washed with 100 c.c. 0.05 per cent cyanide, 100 c.c. 0.025 per cent cyanide and water.

Summary.—The extraction was good using dilute pulp and agitating for 48-hour periods. The cyanide consumption was high; little lime was required.

Amalgamation

Two amalgamation tests, Nos. 5 and 6, were made. The samples were cut at -14 mesh and ground wet in a porcelain ball mill using grey iron balls. Amalgamation was conducted for a period of two hours in pebble jars, using 100 grammes mercury and a pulp density of 1 : 1. The tailings were screened and the products assayed.

Test No.	Size	Weight per cent	Assay gold oz./ton	Content per cent	Per cent of values
5	+ 35	0.6	0.34	0.2	0.5
	- 35+ 48	2.2	0.32	0.6	1.7
	- 48+ 65	12.3	0.52	6.4	15.4
	- 65+100	17.0	0.58	9.9	23.8
	-100+150	18.6	0.56	10.4	25.0
	-150+200	22.2	0.34	7.5	18.0
	-200	27.1	0.24	6.5	15.6
6	+ 65	0.2	0.33	0.1	0.3
	- 65+100	3.8	0.42	1.6	4.8
	-100+150	18.0	0.44	7.9	23.8
	-150+200	43.0	0.32	13.8	41.6
	-200	35.0	0.28	9.8	29.5

	Test No. 5	Test No. 6
Weight of ore, grms.....	1,475	1,475
Degree of crushing, mesh.....	-35	-65
Average value of tailing after amalgamation...	0.42 oz./ton	0.33 oz./ton
Head sample, assay.....	1.10	1.10
Per cent of gold amalgamated.....	61.8	70.0

Summary.—Good results were obtained on the amalgamation of this ore. The gold remaining was largely found in the intermediate sizes. No flowering of the mercury was noticed.

Test No. 7

Flotation at 48 Mesh and Tabling.—One thousand grammes of ore at -14 mesh was ground wet in a ball mill in a dense pulp, 2 : 1, then floated in a Ruth machine. The flotation tailing was tabled on a Wilfley table. The table tailing was screened and the products assayed:—

Product	Weight per cent	Assays		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	15.6	6.16	14.74	86.7	94.6
Table concentrate.....	3.0	1.04	0.42	2.8	0.5
Table tailing.....	56.9	0.17	0.08	8.7	1.9
Slimes.....	24.5	0.08	0.30	1.8	3.0

Flotation reagents—

NaOH.....	2.0 lb./ton added to ball mill.
QED.....	0.05 lb./ton added to Ruth machine.
Pine oil.....	2 drops “ “

NOTE.—Tailing values were taken from results of screen analysis.

Results of Screen Analysis on Table Tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
+ 48 } - 48+ 65 } - 65+100 } - 100+150 } - 150+200 } - 200 }	6.1	0.20	0.05	7.0	3.8
	18.2	0.18	0.03	19.2	6.3
	20.6	0.12	0.04	14.5	10.1
	12.9	0.20	0.06	15.1	10.2
	42.2	0.18	0.13	44.2	69.6

Ratio of concentration..... 5.7 : 1
Loss of original feed..... 5.9 per cent

Summary.—The recoveries by this method of treatment were as follows: gold, 89.5 per cent, of which 86.7 per cent reported in the flotation concentrate; copper, 95.1 per cent, of which 94.6 per cent was in the flotation concentrate. Tabling is of little value, and from the results obtained might easily be dispensed with. The screen analysis revealed that the gold remaining is evenly distributed throughout the sizes, although the 200-mesh material carried the bulk of the copper values.

Test No. 8

Amalgamation and Flotation at 100 Mesh.—One thousand grammes of ore at -14 mesh was ground wet to 100 mesh in a ball mill and amalgamated for two hours with 100 grammes of mercury. The tailing was floated in a Ruth machine and the flotation tailing screened on 100 mesh.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	11.1	1.00	19.10	50.4	94.1
Tailing +100.....	1.3	0.28	0.22	1.8	0.1
Tailing -100.....	87.6	0.12	0.15	47.8	5.8

Head sample..... Gold, 1.10 oz./ton, copper, 2.25 per cent
After amalgamation..... Gold, 0.22 oz./ton
Gold amalgamated, per cent..... 70.9
Copper recovery, per cent..... 94.1

Screen Test on Flotation Tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
+100.....	1.5	0.28	0.22	3.3	2.0
-100.....	98.5	0.12	0.15	96.7	98.0

Average value of tailings..... Gold, 0.12 oz./ton, copper, 0.15 per cent
Flotation reagents—
NaOH..... 2.0 lb./ton added to Ruth machine
QED..... 0.05 lb./ton “ “
Pine oil..... 2 drops “ “

Summary.—The total gold recovery was 89 per cent, of which 70·9 per cent was due to amalgamation. The copper recovery in the flotation concentrate amounted to 94 per cent. Grinding to 100 mesh previous to amalgamation would hardly be feasible in mill operation. Should coarser crushing be tried the values in the flotation concentrates would be raised at the expense of amalgamation recovery.

Test No. 9

Amalgamation and Flotation.—For this test 1,646 grammes of ore at -20 mesh was ground wet to 35 mesh and amalgamated. The amalgamation tailing was floated, the rougher concentrate cleaned, and the tailing screened and assayed.

Product	Weight per cent	Assays		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	6·7	3·40	30·16	46·8	85·2
Flotation middling.....	4·2	1·50	4·54	12·9	8·1
Flotation tailings.....	89·1	0·22	0·18	40·3	6·7

Reagents—

TT mixture..... 0·5 lb./ton added to Ruth machine
No frother was used.

0·1 “ TT was added for cleaning concentrate

The values in the tailing were taken from the screen analysis.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
- 35+ 48.....	2·0	0·32	0·14	2·9	1·5
- 48+ 65.....	13·5	0·28	0·14	17·0	10·4
- 65+100.....	20·8	0·26	0·16	24·2	18·4
-100+150.....	12·6	0·32	0·17	18·1	11·8
-150+200.....	19·7	0·22	0·18	19·5	19·7
-200.....	31·4	0·13	0·22	18·3	38·2

Summary.—Amalgamation recovered 55 per cent of the gold. The flotation concentrate contained 46·8 per cent of the gold remaining after amalgamation. Only 85 per cent of the copper was recovered in the concentrate. The middling was too high both in quantity and grade to be returned to the mill circuit as it would be in actual practice. The tailing contained sufficient gold to warrant cyanidation providing the cyanide consumption was moderate.

Test No. 10

Amalgamation, Flotation, and Cyanidation.—One thousand grammes of ore at -14 mesh was ground wet in a ball mill to 48 mesh and amalgamated for 1 hour in a 1 : 1 pulp using 100 grammes of mercury. The amalgamation tailing was dewatered and floated. The flotation tailing

was sampled and cyanided for 65 hours in a 1 : 2 pulp with cyanide strength of 0.075 per cent. The cyanide tailing was screened.

Head sample before amalgamation..... 1.10 oz./ton
 After amalgamation..... 0.32 "
 Gold amalgamated..... 70.8 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	9.0	1.58	21.38	43.8	89.4
Flotation tailing.....	91.0	0.20	0.25	56.2	10.6

Screen analysis of cyanide tailing (605.7 grammes).

Mesh	Weight per cent	Assay Au oz./ton	Per cent of values
- 48+ 65.....	11.1	0.18	27.8
- 65+100.....	14.2	0.12	23.6
-100+150.....	15.0	0.09	19.4
-150+200.....	26.0	0.03	11.1
-200.....	33.7	0.04	18.1

Test No. 11

Amalgamation, Flotation, and Cyanidation.—Procedure as in test No. 10, except that the flotation tailing was ground to -200 mesh before cyanidation. The grinding was done in a ball mill, the discharge being screened and the oversize returned to the mill.

Head sample before amalgamation..... 1.10 oz./ton
 After amalgamation..... 0.33 "
 Gold amalgamated..... 70.0 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	7.9	1.15	21.62	27.5	77.4
Flotation tailing.....	92.1	0.23	0.54	72.5	22.6

Test No. 12

Amalgamation, Flotation, and Cyanidation.—Procedure as in test No. 11, except that amalgamation tailing was ground to -200 mesh for flotation:—

Head sample before amalgamation..... 1.10 oz./ton
 After amalgamation..... 0.30 "
 Gold amalgamated..... 72.7 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	12.4	1.42	14.40	58.8	83.9
Flotation tailing.....	87.6	0.14	0.39	41.2	16.1

Results of Cyanidation, Tests 10, 11, and 12—

Test No.	Weight grms.	Solution c.c.	KCN per cent	Assay		Extraction per cent	Consumption lb./ton		Time hours
				Heads	Tails		KCN	CaO	
10	605.7	1195	0.075	0.20	0.072	64.0	4.76	2.2	65
11	611.2	1789	0.050	0.28	0.060	78.6	6.50	3.9	47
12	611.7	1788	0.10	0.14	0.040	71.5	7.40	3.9	47

Details of Tests—

Test No.	Time	KCN per cent	CaO per cent	KCN raised to	Remarks
10	17	0.014	0.023	0.075	
	24	0.066	0.023	No KCN added.
	41	0.046	0.022	0.075	
	47	0.066	0.021	No KCN added.
11	65	0.042	0.019	Ceased agitation.
	17	0.006	0.009	0.050	0.5 grm. CaO added.
	24	0.040	0.021	
	41	0.016	0.019	0.05	0.1 grm. CaO added.
12	47	0.040	0.018	Ceased agitation.
	17	0.042	0.015	0.10	0.2 grm. CaO added.
	24	0.066	0.018	0.2 grm. CaO added.
	41	0.038	0.02	0.10	
	47	0.092	0.022	Ceased agitation.

In each test the cyanide tailing was washed with 200 c.c. 0.10 per cent cyanide; 200 c.c. 0.05 per cent cyanide; and 400 c.c. water.

Summary Tests 10, 11, and 12

The amalgamation in each test was good. About 70 per cent of the gold was extracted at 48 mesh.

Flotation—Gold.—The best flotation results of gold were obtained in test No. 12, where the operation was conducted at -200 mesh. In this test more bulk was removed in the concentrate with a corresponding lower grade of copper. *Copper.*—The flotation results in all three tests were poor, the best recovery being 89.4 per cent in test No. 10. The flotation reagents used in each test were:—

Sodium carbonate.....	3.0 lb./ton
QED.....	0.5 "
Pine oil, G.N.S. No. 5.....	0.10 "

Cyanidation.—Extraction of the gold from the flotation tailings varied from 64 to 79 per cent. The 64 per cent was obtained on tailings cyanided at 48 mesh in test No. 10. In this case the coarser sizes contained considerable gold. The cyanide consumption varied from 4.8 pounds to 7.4 pounds per ton. The high copper in the flotation tailings is largely responsible for this high consumption.

Test No. 14

Amalgamation, Flotation, and Cyanidation.—Procedure as in test No. 11.

Head sample before amalgamation.....	1.10 oz./ton
After amalgamation.....	0.32 "
Gold amalgamated.....	70.4 per cent

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	10.6	1.21	17.80	39.4	90.2
Flotation tailing.....	89.4	0.22	0.23	60.6	9.8

Reagents—	
Water-gas tar, coal-tar creosote, 50 per cent mixture.....	0.7 lb./ton
Pine oil.....	0.1 "
Cyanide results—	
Flotation tailings, weight.....	593.5 grms.
Solution.....	1200 c.c.
Cyanide.....	0.05 per cent
Assay—	
Heads.....	0.22 oz./ton
Tails.....	0.03 "
Extraction.....	86.4 per cent
Reagents consumed—	
KCN.....	4.8 lb./ton
CaO.....	4.5 "
Time.....	48 hours
Original head sample.....	1.10 oz./ton
Final tailing.....	0.03 "
Total extraction.....	97.3 per cent

Summary

Amalgamation recovered 70.4 per cent of the total gold.

Flotation.—The copper recovery was greatly improved over previous attempts, but the gold recovery was low, only 39.4 per cent reporting in the concentrate.

Cyanidation.—Gold extraction 86.4 per cent was obtained using less than 5 pounds per ton cyanide.

Test No. 15—Flotation and Cyanidation

One thousand grammes of the ore at -14 mesh was ground wet to 48 mesh and floated in a Ruth machine. The concentrate was cleaned and the tailings were reground to -200 mesh, sampled, and cyanided.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Concentrate.....	6.6	12.50	30.06	68.3	90.3
Middling.....	2.0	5.97	2.96	9.8	2.7
Tailing.....	91.4	0.29	0.17	21.9	7.0

Reagents—			
Lime.....	5.0	lb./ton added to ball mill	
Xanthate.....	0.07	" " cells	
Pine oil.....	0.10	" " "	
Cyanidation—			
Flotation tailing.....	599.4	grms.	
Solution.....	1200	c.c.	
Cyanide.....	0.05	per cent	
Heads, gold.....	0.29	oz./ton	
Tailing, gold.....	0.025	"	
Extraction.....	91.4	per cent	
Cyanide consumed.....	4.0	lb./ton	
Lime consumed.....	2.9	"	
Time.....	48	hours	
Original head sample.....	1.21	oz./ton Au	
Final tailing.....	0.025	"	
Total extraction.....	98	per cent	

Summary.—The results of the flotation test were the best yet obtained. The concentrate carried 68.3 per cent of the gold and 90.3 per cent of the copper, and the middling is more commensurate with what would be obtained in large-scale practice. The tailing contained 0.29 ounce gold per ton and 7.0 per cent of the total copper. The cyanidation results are good, 91.4 per cent of the gold in the tailing goes into solution, using 4.0 pounds per ton cyanide.

GENERAL SUMMARY

Cyanidation.—In test No. 4, 95 per cent of the gold is dissolved by 0.05 per cent cyanide solution in a pulp density of 1:3 when agitated for 48 hours. The cyanide consumption is high, 11 pounds per ton, an amount which would almost prohibit this type of treatment. Our experience on a similar ore has shown that on repeating the operation using the fouled solution, the cyanide consumption will decrease with each succeeding cycle, but the extraction also decreased. In addition to the lessened dissolving power of the solution, incomplete precipitation might reasonably be expected. This is due to the presence of copper in the solution. Strengthening the solution just previous to precipitation and the addition of an excess over the usual practice of lead acetate will ameliorate this condition. It might, in mill practice, pay to run to waste say one-fifth of the solution after precipitation, which would not allow the copper to accumulate indefinitely in the mill circuit.

Amalgamation.—Amalgamation was satisfactory. The recovery by this method of treatment varied from 55 per cent at -35 mesh in test No. 9, to 70 per cent at -48 mesh in tests Nos. 10, 11, and 12. Grinding finer to 100 mesh as in test No. 8 produced little better results. Accordingly amalgamation could be inserted with profit in the mill flow-sheet.

Flotation.—Flotation can be applied with good results to this ore. It may be used as a primary step where 90 per cent of the copper and 68 per cent of the gold may be obtained in a high-grade concentrate which could be shipped to a smelter, or it might be inserted after amalgamation, should this be installed. In either case, the tailing from the cells contains sufficient gold to warrant cyanidation. In practice an 0.12 per cent copper tailing could be obtained. On such a tailing less than 4 pounds cyanide per ton would be sufficient. It would also eliminate possible precipitation troubles. The flotation tailing when ground to -200 mesh is more readily cyanided with a lower final tailing, 0.025 ounce gold per ton.

CONCLUSIONS

From the results obtained in the above tests, amalgamation, flotation, and cyanidation of the reground flotation tailing would recover 98 per cent of the gold and 90 per cent of the copper. Possibly the concentrate could be stacked pending better conditions as to freight rates, price of copper, or the erection of a smelter nearby.

Report No. 225

THE RECOVERY OF FOUNDRY METAL FROM SWEEPINGS

J. S. Godard

Shipments.—A shipment of 400 pounds of sweepings was received February 7, 1925, from the Canada Bronze Co., Montreal, Que.

Characteristics of the Sweepings.—The material consisted of sweepings remaining after casting, and was composed of gates, spills, etc., from the castings, as well as pieces of coal, and considerable sand from the moulds, all varying in size from 2 inches to a very fine material, 100 mesh, and analysed approximately, copper 30 per cent, lead 10 per cent, tin 3.5 per cent, zinc 7 per cent.

Sampling and Analysis.—It was almost impossible to obtain a satisfactory head sample owing to the nature of the material. The copper in the head sample will be calculated from the products of the various tests.

Purpose of Experimental Work.—The purpose of these tests was to raise the metallic content of the material, particularly the copper, by grinding and gravity concentration, to such a point as would permit of its being re-melted and re-cast. Formerly the material was shipped to a U.S. smelter, but high freight rates made it advisable to find a simple and cheap method of treatment on the company's premises. The company was desirous of recovering the copper which was the predominating metal and as they already had a Wilfley table they desired, if possible, a process which might employ tabling.

EXPERIMENTAL TESTS

Test No. 2

Grinding, Screening, and Tabling.—A sample of 20 pounds was cut from 300 pounds of the sweepings crushed in rolls and screened on a 4-mesh screen. The +4-mesh product was a good grade of metallics. From the -4 mesh a sample of 3,887 grammes was cut with a Jones sampler. This was crushed once in the rolls and screened on 48 mesh. The +48-mesh product was ground wet in a ball mill and again screened on 48 mesh. The +48-mesh product was a good grade of metallics and was not further treated. The -48 mesh from the rolls was tabled on a small Wilfley table, re-running the middlings to make a concentrate and tailing. The -48 mesh from the ball mill was treated similarly and the slimes from both tabling operations were combined.

Results.—

1. Sampling, grinding, and screening—

Oversize from 20 pounds crushed to 4 mesh.....	329.5 grms.
Sample cut from -4 mesh.....	3,887.0 “
+48 mesh after passing once through rolls.....	1,913.0 “
-48 mesh after passing once through rolls.....	1,974.0 “

2. Concentration—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+48 mesh from ball mill.....	565.0	16.5	74.05	58.6
Table concentrate -48 mesh from ball mill.....	228.0	6.6	52.02	16.6
Table concentrate -48 mesh from rolls.....	351.2	10.2	23.04	11.3
Table tailing -48 mesh from ball mill.....	408.7	11.9	6.14	3.5
Table tailing -48 mesh from rolls.....	1,242.5	36.2	3.12	5.5
Slimes from both tablings.....	635.5	18.6	5.01	4.5

Head sample from products, omitting the +4 mesh, (copper).....	20.72 per cent
Total feed.....	3,887 grms.
Products recovered.....	3,431 “
Loss in slimes.....	456 “
Loss of original feed.....	11.7 per cent
Value in the concentrates.....	86.5 “

Summary.—The loss from sliming indicated the tendency of the material to slime when treated in this way. The concentrate on 48-mesh screen after grinding in ball mill is very good. That produced from tabling the -48 mesh from the rolls was the poorest of the three. The table showing results of concentration is given below.

Product	Weight		Analysis Cu per cent	Per cent of values
	Grms.	Per cent		
+48 mesh from ball mill.....	565.0	49.4	74.05	67.7
Table concentrate -48 mesh from ball mill.....	228.0	19.9	52.02	19.2
Table concentrate -48 mesh from rolls.....	351.2	30.7	23.04	13.1

Average copper content in concentrates..... 54.0 per cent

Test No. 4

Grinding in Ball Mill, Screening on 35 Mesh, and Tabling the -35 Mesh.—A sample of 15 pounds was cut from 180 pounds of the sweepings. This was screened on 4 mesh. The oversize was passed once through rolls and again screened on 4 mesh. From the -4 mesh a sample of 1,562 grammes was cut and this was ground wet in a ball mill for 15 minutes and screened on 35 mesh. The oversize was reground for a further 15 minutes and again screened on 35 mesh. The oversize from this was a good metallic product. The -35 mesh product was tabled on a Wilfley table, re-running the middlings and making a final concentrate, middling, and tailing. The slimes from the screenings were kept separate from those of the table.

Results.—

1. Sampling, grinding, and screening—

Oversize from 15 pounds crushed to 4 mesh.....	277.7 grms.
Sample cut from -4 mesh.....	1,562.0 "

2. Concentration:—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+35 mesh screen.....	341.4	22.2	62.5	69.0
Table concentrate -35 mesh.....	360.7	23.5	22.08	25.8
Table middling -35 mesh.....	40.0	2.6	2.38	0.3
Table tailing.....	599.7	39.1	1.14	2.2
Slimes (1) screen.....	110.7	7.2	4.52	1.6
Slimes (2) table.....	82.7	5.4	3.99	1.1
<i>Table of concentrates—</i>				
+35 mesh.....	341.4	48.6	62.50	72.8
Table concentrates.....	360.7	51.4	22.08	27.2

Average copper content of concentrates.....	41.73 per cent
Original feed.....	1,562.0 grms.
Products from operations.....	1,535.2 "
Loss (slimes from table).....	26.8 "
Loss of original feed.....	1.7 per cent
Values in copper concentrates.....	94.8 "

Test No. 5

Grinding, Screening, and Tabling.—A sample of 1,869.5 grammes was cut from 160 pounds. This was screened dry on 35 mesh, and the oversize reground 15 minutes in a ball mill and again screened; this operation was twice repeated. The final +35 mesh was a good grade of metallics. The -35 mesh was tabled re-running the middlings to make a concentrate and tailing. The slimes from the screen were kept separate from those from the table.

Results—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+35 mesh concentrate on screen.....	437.4	24.1	60.70	67.3
Table concentrate -35 mesh.....	414.9	22.8	25.00	26.2
Table tailing -35 mesh.....	778.4	42.8	2.19	4.3
Slimes (1) screen.....	55.2	3.1	5.06	0.7
Slimes (2) table.....	131.0	7.2	4.58	1.5
<i>Table of concentrates—</i>				
+35 mesh.....	437.4	51.3	60.7	71.9
Table concentrates.....	414.9	48.7	25.0	23.1

Average copper content of concentrates.....	43.31 per cent
Average head sample from products, (copper).....	21.75 per cent
Original feed.....	1,869.5 grms.
Products, total.....	1,816.9 "
Loss.....	52.6 "
Loss of original feed.....	2.8 per cent
Recovery in concentrate (of the copper).....	93.5 "

Test No. 6

Grinding, Screening, and Tabling.—A sample of 1,795 grammes was cut from 150 pounds, and crushed dry in rolls, then ground wet in a ball mill for 20 minutes and screened on 65 mesh. The oversize was again ground in a ball mill and screened. These operations were twice repeated. The oversize from the last screening was a good grade of metallics. The -65-mesh material from the screening was tabled, re-running the middlings and making a final concentrate, middling, tailing, and slimes.

Results.—

Product	Weight		Analysis Cu per cent	Per cent of Cu values
	Grms.	Per cent		
+65 mesh.....	360.4	21.1	74.20	71.6
Table concentrate.....	185.7	10.9	37.02	18.5
Table middling.....	25.7	1.5	12.63	0.9
Table tailing.....	907.0	53.1	2.53	6.1
Table slimes.....	228.0	13.4	4.78	2.9
<i>Table of concentrates—</i>				
+65 mesh.....	360.4	66.0	74.20	79.6
Table concentrates.....	185.7	34.0	37.02	20.4

Average head sample from products (copper).....	21.86 per cent
Original feed.....	1,795.0 grms.
Products.....	1,706.8 “
Loss.....	88.2 “
Loss due to slimes (of original feed).....	4.9 per cent
Recovery in concentrates (copper).....	90.1 “
Average copper content of concentrate.....	61.56 “

Summary.—In the above tests, good recoveries were made by grinding wet in a ball mill, screening out the metallics, and tabling the undersize from the screen. The best results were obtained on material ground to -65 mesh.

Recommendations.—An installation consisting of a 3-foot 8-inch cylinder Hardinge ball mill, maximum capacity 1 ton per hour, is recommended. This could be operated in conjunction with the Wilfley table already in possession of the company. Fifteen horsepower will be sufficient to operate both the ball mill and the table.

Grinding to 65 mesh will be sufficient to free the copper, the -65-mesh material being fed to the table. The +65-mesh product can be obtained by emptying the ball mill about once per week.

Report No. 226**AMALGAMATION AND CONCENTRATION OF A MATACHEWAN
GOLD ORE****C. S. Parsons and J. S. Godard**

Shipment.—A shipment of 16,312 pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories, March 26, 1925, from the Porcupine Goldfields Development and Finance Corporation, Matachewan. The ore was from the Young-Davidson property, Powell township, Ontario.

Purpose of Experimental Work.—The erratic distribution of the free gold in this ore made underground sampling unreliable. A mill run was desired to check the results obtained by underground sampling, and to obtain data and determine a method for the recovery of the gold.

Characteristics of the Ore.—The ore consisted of a hard, red granitic porphyry, mineralized with iron pyrite and containing fine free gold. The gold values were very spotty and it was difficult to obtain check assays from the samples.

Sampling and Analysis.—The sampling was given careful attention. Six samples in all were cut from the shipment. The ore was crushed in a jaw crusher and rolls to $\frac{1}{2}$ -inch, passed through a Vezin sampler which cut out a one-tenth part. This was reduced to 10 mesh and then coned and quartered. Each quarter was carried down by graded crushing and splitting, giving four separate head samples. Two additional samples were taken as a re-check on the four, by reducing the rejected half of the first tenth out of the Vezin and reducing it by graded crushing and splitting in a Jones riffle sampler. The results were as follows:—

Sample No. 1.....	0.140 oz./ton gold
No. 2.....	0.141 " "
No. 3.....	0.16 " "
No. 4.....	0.132 " "
No. 5.....	0.145 " "
No. 6.....	0.130 " "
Average.....	0.141 " "

Duplicate assays made of each of these samples were either the same or showed a difference of 0.01 and not exceeding 0.02 ounce per ton.

Stamp Battery, Amalgamation and Concentration

The ore was fed to a 5-stamp battery carrying a 40-mesh screen. The pulp passed over amalgamation plates to recover the gold. The tailing from the plates was run direct to a standard deck Wilfley table without classification of any kind. The table under this condition did not do good work, as it was operated above capacity and the fine sulphide lay in the riffles under the coarse sand and escaped into the tailing. The tailing was pumped to a tank for settling to await further treatment. In order to recover the fine sulphides the tailing was passed over a 60-mesh Callow belt screen, the undersize going direct to the table. The fines represented 85 per cent by weight, and the coarse 15 per cent. Had the concentration been made on a classified feed, a cleaner and higher grade concentrate would have been produced. The sample of the tailing from the first tabling was taken between the table and the pump before the pulp was pumped to the settling tank. The content in the sample plus the content in the concentrate checked with the amalgamation tailing, which was the feed to the table, showing that the sampling was accurate. In settling this table tailing, there was a loss of fine slimes and probably some fine concentrate in the overflow, so that the accuracy of the final tailing samples, which showed 0.02 ounce gold per ton and 0.012 ounce per ton, is questionable. It would, therefore, be more accurate to take the content of gold in the concentrate obtained and subtract it from the original amalgamation tailing in order to arrive at the values for the final table tailing. This would give a value of 0.0248 ounce per ton representing a loss of 16.61 per cent of the gold values.

The first concentrate contained considerable gangue and was fairly coarse, whereas the concentrate from the -60-mesh material was fine, but much cleaner. The first concentrate contained 0.56 ounce per ton and the second 0.36 ounce per ton. This difference must be due to the gold values in the coarse pyrite not being amalgamated to the same extent as those from the fine pyrite.

SUMMARY

Crushing—

Stamp battery.....	Five 1,250-pound stamps
Height of drop.....	6 inches
Number of drops.....	95 per minute
Height of chuck block.....	4½ inches
Screen.....	40 mesh, diagonal slots
Stamp duty.....	3.2 tons per stamp per 24 hours

With a greater drop the stamp duty would be increased, probably to 4 tons per stamp per 24 hours.

Amalgamation—

Apron plates.....	45.0 square feet
Lip plates.....	2.33 "
Chuck block plate.....	2.0 "
Recovery in fine bullion.....	0.63 ounce or 54.8 per cent
Recovery, figuring from assay of amalgamation tailing.....	61.05 "

The latter recovery will be more correct as it is impossible to scrape all the amalgam from the plates.

Ore to battery, weight.....	16,312.0	pounds
Assay, gold.....	0.141	oz./ton
Content, gold.....	1.15	ozs.
Clean up of battery, weight.....	67.5	lb.
Assay.....	0.65	oz./ton
Content.....	0.022	ozs.
Amalgamation tailing, weight (by calculation).....	16,244.5	lb.
Assay.....	0.0525	oz./ton
Content.....	0.426	oz.
Fine gold bullion recovered from inside, and from plates...	0.63	oz.
Recovered as bullion.....	54.8	per cent
Recovery as figured from tailing assay.....	61.06	"
Gold absorbed by battery plates.....	0.235	oz.
Per cent absorbed by battery plates.....	6.26	

Concentration—

Concentrate from first tabling of amalgamation tailing.....	525.5	pounds
Assay.....	0.68	oz./ton
Content.....	0.179	oz.
Per cent recovered from tailing.....	42.0	
Per cent of total gold values in heads.....	15.5	
Tailing from first tabling, by calculation.....	15,719.0	pounds
Assay.....	0.03	oz./ton
Content.....	0.236	oz.
Per cent of gold.....	55.4	
Per cent of total gold.....	20.5	
Concentrate from second tabling, -60 mesh.....	270.0	pounds
Assay.....	0.36	oz./ton
Content.....	0.0486	oz.
Tailing from second tabling.....	13,091.0	pounds
Assay.....	0.012	oz./ton
Content.....	0.0785	oz.
Concentrate, +60 mesh.....	42.5	pounds
Assay.....	0.36	oz./ton
Content.....	0.00765	oz.
Tailing, +60 mesh.....	2,315.5	pounds
Assay.....	0.02	oz./ton
Content.....	0.0232	oz.

Recapitulation of Concentration Results—

Product	Weight lb.	Assay oz./ton	Content oz.	Per cent of Au values
First concentrate.....	525.5	0.68	0.179	15.54
-60 concentrate.....	270.0	0.36	0.0486	4.23
+60 concentrate.....	42.5	0.36	0.0076	0.66
Total concentrate.....	838.0	0.56	0.235	20.43
-60 tailing.....	13,091.0	0.012	0.0785	6.83
+60 tailing.....	2,315.5	0.02	0.0232	2.02
Total tailing.....	15,406.5	0.013	0.1017	8.85

Summary of Concentration Results—

Table concentrates, weight.....	838.0	pounds
Percentage by weight in table concentrates.....	5	
Assay table concentrates.....	0.56	oz./ton
Content in table concentrates.....	0.235	oz.
Per cent of gold values in table concentrates.....	20.43	
Table tailing, weight.....	15,406.5	pounds
Calculated content in table tailing.....	0.191	oz.
Calculated assay of table tailing.....	0.0248	oz./ton
Per cent of gold values in tailing.....	16.61	

Balance—

Product	Content oz.	Per cent
Amalgamated.....	0.702	61.05
Clean up battery.....	0.022	1.91
Concentrates.....	0.235	20.43
Tailings.....	0.102	8.85
Totals.....	1.061	92.24
Feed to battery.....	1.150	
Discrepancy.....	0.089	7.76
Gold recovered by amalgamation and in table concentrates..		81.47

Cyanidation of Table Concentrates.—A number of cyanidation tests were made on the table concentrates. A summary of the results obtained is given in brief form in the following table.

Remarks.—The concentrate was ground wet and intermittently screened through the required mesh. The grinding was done with pebbles, and lime equivalent to 10 pounds per ton was added to the pebble mill during grinding.

In the first three tests the grinding was through 150 mesh, and in the last two tests, through 200 mesh. Finer grinding and longer contact increased the cyanide consumption, but the increase in extraction was over 10 per cent.

Summary of Cyanide Tests on Pyrite Concentrates from Amalgamation Tailings

Test	Mesh	Weight grms.	Solution c.c.	Time of agitation	KCN per cent	Assay oz./ton		Extraction per cent	Reagents consumed	
						Heads	Tails		KCN	CaO
				hours					lb./ton	lb./ton
A	150	489.0	1511	48	0.15	0.56	0.14	75.0	2.53	3.7
B	150	289.0	1213	48	0.20	0.56	0.12	78.8	3.1	2.5
C	150	241.0	1259	48	0.10	0.56	0.10	82.2	2.4	10.3
D	200	512.0	2500	68	0.075	0.56	0.06	89.2	4.34	
E	200	513.5	2500	68	0.150	0.56	0.06	89.2	4.85	

Report No. 227

THE PREPARATION OF VOLCANIC ASH FROM WALDECK, SASKATCHEWAN,
FOR INDUSTRIAL PURPOSES

R. K. Carnochan

Shipments.—Two bags of volcanic ash, net weight $192\frac{1}{2}$ pounds, were received January 8, 1925; and two bags, net weight 95 pounds, January 23, 1925, at the Ore Dressing and Metallurgical Laboratories. Both shipments were submitted by Mr. W. C. Vance, of the Van Kel Cleaners, Limited, Swift Current, Sask., and were taken from the company's volcanic ash deposit near Waldeck.

Characteristics of Volcanic Ash Submitted—

Size.....Pieces broken to about 4 inches.
 Colour.....Light buff.
 Weight.....Very light.
 Texture.....Fine-grained; the grains were not bonded together solidly; a powder could be produced by rubbing with the fingers; the ash could readily be cut with a knife.

Purpose of Experimental Tests.—Tests were desired to determine possible methods of crushing, grinding, and classification, to produce various grades of products, which could be used in oil filtering, as well as hand cleaner, household cleanser, and metal polish.

Experimental Tests on Shipment No. 1.—After selection of a few specimens, the remaining 192 pounds of shipment No. 1 was crushed by means of a jaw crusher and rolls to all pass an 8-mesh screen. The -8-mesh material was used in making grinding tests with a Raymond grinder in which an effort was made to grind to almost all pass -48 mesh, and have as much as possible +100 mesh. The ground product from the Raymond was put through a Gayco air separator so as to separate it into -48+100 and -100-mesh products.

SUMMARY

The tests made on shipment No. 1 show:—

1. The ash crushed very easily in the jaw crusher.
2. Rolls are unsuited for the crushing of this material, as the ash shows a tendency to form cakes which do not go through the screen or crush up when re-run through the rolls.
3. The Raymond grinder can be adjusted to take material -8 mesh and give a product:—

+ 48 mesh.....	0.2 per cent
- 48+100 ".....	26.8 "
-100 ".....	73.0 "

4. The Gayco air separator gives a good fine product:—

+100 mesh.....	5.4 per cent
-100 ".....	94.6 "

The coarse product is not very good, as it tests:—

+100 mesh.....	24.8 per cent
-100 ".....	75.2 "

The trouble in the operation of this separator is that the ash keeps breaking up as it is going through, so that there are always fines in the coarse product. When this product is re-run five times, as was done to clean the fines out, a very small amount of coarse material remains.

Experimental Tests on Shipment No. 2.—Seventy pounds was crushed in a jaw crusher to $1\frac{1}{2}$ inch, and then crushed in a hammer mill to all pass a 6-mesh screen. The hammer mill product was screened and gave:—

- 6+10 mesh.....	18 pounds
-10+48 "	24 "
-48 "	24 "

The -10+48 was crushed in a Raymond grinder to nearly all pass 48 mesh and have as much as possible +100 mesh. The product from the Raymond mill was mixed with the -48-mesh material from the screen, and the mixture separated in a Gayco air separator into:—

Oversize.....	0.5 pounds
Fines.....	46.5 "

A screen test on the oversize shows:—

+ 48 mesh.....	18.4 per cent
- 48+100 "	41.4 "
-100 "	40.2 "

A screen test on the fines shows:—

+ 48 mesh.....	2.5 per cent
- 48+100 "	6.4 "
-100 "	91.1 "

SUMMARY

1. The hammer mill is well adapted to the crushing of the ash.
2. The Raymond grinder takes the -10+48-mesh product from the screen and gives a product:—

+ 48 mesh.....	7.8 per cent
- 48+100 "	7.6 "
-100 "	84.6 "

3. As in the test on shipment No. 1, the Gayco air separator gave a good fine product. The coarse product was better than that obtained from shipment No. 1, as it tested:—

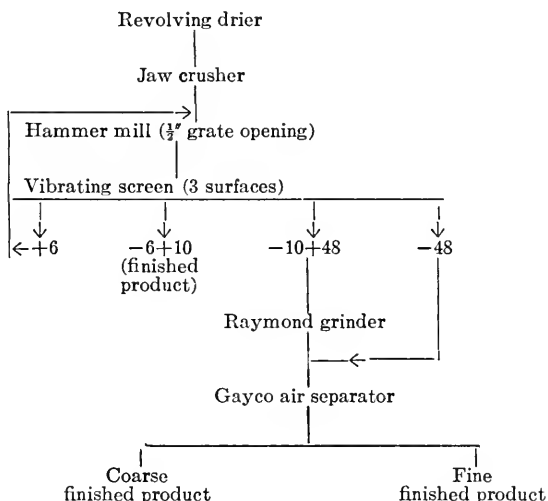
+ 48 mesh.....	18.4 per cent
- 48+100 "	41.4 "
-100 "	40.2 "

This is 59.8 per cent +100 mesh, whereas the coarse from shipment No. 1 is only 24.8 per cent +100 mesh. This improvement is due to the fact that the proper setting of the machines had been determined when working on the first shipment.

The ash keeps breaking up when put through the Gayco air separator, and this results in a very small amount of coarse being produced. The Gayco air separator used was a small one, and in commercial practice where a larger machine would be used, one pass instead of five as used in these tests, would be sufficient to secure a coarse product free of fines.

CONCLUSIONS

1. The following flow-sheet is recommended for the dry crushing of the volcanic ash:—



2. Small tests made on the volcanic ash show that it is very porous and absorbs moisture, and that it is consequently very difficult to dry, hence the drier installed should be large enough to handle a material of this nature at the required capacity.

3. The above flow-sheet would give a $-6+10$ -mesh product suitable for oil filtering, a coarse product from the air separator suitable for making a hand cleaner, and a fine product for making household cleanser. The tests on shipment No. 2 show that these products would be produced in the following proportions:—

$-6+10$ mesh.....	18.0 pounds	25.7 per cent
Air separator, coarse.....	0.5 "	0.7 "
Air separator, fines.....	51.5 "	73.6 "
Feed.....	70.0 "	100.0 "

In the above table it is assumed that the dust loss in the tests would be fines, and that in regular mill work only a small amount would be lost in dust. In regular practice with a large air separator, more coarse material would be produced and less fines, as only one pass would be needed.

4. If a very fine product was desired for metal polish, or any other purpose, a part of the fines from the air separator could be re-run, or put through a second air separator, so as to give a very fine product.

Report No. 228

THE CONCENTRATION OF A DRY SILVER ORE FROM THE SLOCAN SILVER MINES LTD., ALAMO, B.C.

C. S. Parsons

Shipment.—A shipment of 200 pounds of silver-lead ore was received May 7, 1925, from the Slocan Silver Mines, Alamo, B.C. The shipment consisted of two sacks designated as 'Lot No. 1', one sack as 'Lot No. 2,' and one sack as 'Lot No. 3'.

Characteristics and Analysis of Shipment.—The three lots represented distinct types of ores. Lot No. 1 consisted of clean siliceous ore representing 90 per cent of the minable ore. Lot No. 2 consisted of badly oxidized ore, containing a large amount of brown manganese oxide, representing 2 per cent of the minable ore. Lot No. 3 consisted of ore found near the breaks in the vein in which secondary minerals have formed, and represented 8 per cent of the minable ore. The analysis of the samples showed them to contain:—

	Lead per cent	Zinc per cent	Copper per cent	Silver oz./ton
Lot No. 1.....	1.05	0.80	0.27	37.7
Lot No. 2.....	2.61	0.75	2.80	349.6
Lot No. 3.....	0.65	0.75	1.14	150.3

Purpose of Experimental Tests.—The experimental tests were conducted to determine the recovery and grade of concentrate that could be obtained from each type of ore by flotation, and from the ores when mixed in the proportions given above, so that the owners might determine whether it would be more profitable to ship the ore direct to the smelter, or concentrate it to obtain a high-grade shipping product.

EXPERIMENTAL TESTS

Procedure.—A series of tests were made on each of the three lots, and on a composite sample made up in the proportion of the percentages of each in the run-of-mine. The sample cut out from each lot and from the composite lot for test purposes, was assayed for silver content only. The results are given in the following table:—

Results of Flotation Tests:—

Lot No.	Test No.	Product	Weight		Assay Ag. oz./ton	Values Ag. per cent	Remarks
			Grms.	Per cent			
1	1	Head sample.....	26.70	3 per cent +100 mesh; pulp neutral. Xanthate 0.4 lb./ton. Pine oil to froth.
		Concentrate.....	89.7	9.0	271.30	91.0	
		Tailing.....	901.5	91.0	2.65	9.0	
		Totals.....	991.2	100.0	26.9	100.0	
	2	Concentrate.....	110.5	11.1	220.60	91.2	3 per cent +100 mesh; pulp neutral. Soda ash 2 lb./ton, xanthate 0.4 lb./ton. Pine oil to froth.
		Tailing.....	883.9	88.9	2.65	8.8	
		Totals.....	994.4	100.0	26.8	100.0	
	3	Concentrate.....	50.9	5.6	420.60	89.4	3 per cent +100 mesh; pulp neutral (40 per cent coal tar, 60 per cent coal-tar creosote) 0.75 lb./ton, 0.2 lb./ton xanthate. Pine oil to froth.
		Tailing.....	861.2	94.4	2.96	10.6	
		Totals.....	912.1	100.0	26.3	100.0	
2	1	Head sample.....	233.80	—100 mesh; pulp neutral (40 per cent coal tar, 60 per cent coal-tar creosote) 0.75 lb./ton, 0.2 lb./ton xanthate. Pine oil to froth.
		Concentrate.....	117.7	11.7	1600.90	78.9	
		Tailing.....	884.0	88.3	56.88	21.1	
		Totals.....	1001.7	100.0	238.0	100.0	
	1	Head sample.....	113.10	3 per cent +100 mesh; pulp neutral (40 per cent coal tar, 60 per cent coal-tar creosote) 1 lb./ton, 0.2 lb./ton xanthate. Pine oil to froth.
3		Concentrate.....	81.2	8.0	1086.80	77.8	
		Tailing.....	930.3	92.0	27.14	22.2	
		Totals.....	1011.5	100.0	112.2	100.0	

Composite Sample: Lot No. 1, 90 per cent; No. 2, 2 per cent; No. 3, 8 per cent; Heads Ag. 37.75 oz.

Test No.	Product	Weight		Assay Ag. oz./ton	Values Ag. per cent	Remarks
		Grms.	Per cent			
1	Concentrate.....	91.2	9.0	375.00	88.3	3 per cent +100 mesh; pulp neutral. (40 per cent coal tar, 60 per cent coal-tar creosote) 0.75 lb./ton, 0.2 lb./ton xanthate. Pine oil to froth.
	Tailing.....	924.0	91.0	4.94	11.7	
	Totals.....	1015.2	100.0	38.2	100.0	

Screen Test—Lot No. 1—Feed to Flotation—

Mesh	Weight, grms.	Weight, per cent
+ 65.....	1.2	0.24
— 65+100.....	13.4	2.63
—100+150.....	66.4	13.28
—150+200.....	75.2	15.04
—200.....	343.8	68.76

SUMMARY AND CONCLUSIONS

The tests show that there is no difficulty in concentrating by flotation the ore of Lot No. 1, which represents 90 per cent of the run-of-mine. The recovery was high, and in practice should exceed 90 per cent of the silver values. It is rather difficult to determine the maximum grade of concentrate that could be obtained in mill operations, from the results of these small-scale tests, but as the concentrates obtained in tests Nos. 1, 2, and 3, were not recleaned, and are only rougher concentrates, it would seem that a concentrate exceeding 500 ounces silver to the ton could be obtained in actual plant operations. The best results were obtained in a neutral pulp with the use of a mixture of coal tar and coal-tar creosote (acid) and potassium xanthate. The coal tar and creosote mixture was added to the ball mill and the xanthate in solution direct to the head of the flotation cells.

The results of the tests on Lots Nos. 2 and 3 show a low recovery. No improvement is shown in the results on the composite sample. In fact, if the recovery made on Lot No. 1 is taken as an average of 90.5 per cent, on Lot No. 2 as 78.9 per cent, and on Lot No. 3 as 77.8 per cent, the calculated recovery on the composite sample should be 89.3 per cent instead of 88.3 per cent as obtained. This checks very closely, showing that the total recovery obtained by concentrating each type of ore separately is practically the same as when the ores are mixed and concentrated together.

From the results of the above tests, it would seem that the concentration by flotation of the ore represented by Lot No. 1 would be a profitable operation, and it would also seem that the ores represented by Lots Nos. 2 and 3 should be shipped direct.

Report No. 229

THE CONCENTRATION OF A COPPER ORE FROM THE PITT MINING CO.,
PITT LAKE, B.C.

J. S. Godard

Shipments.—A shipment of copper ore, gross weight 218 pounds, was received on May 23, 1925, from the Pitt Mining Co., Pitt Lake, B.C. The shipment represented the low-grade ore from the company's property.

Characteristics of the Ore.—The ore is chalcopyrite associated with pyrite in a quartz gangue. Some silver and a small amount of gold is associated with the sulphides.

Purpose of Experimental Tests.—The purpose of the tests was to concentrate the copper and silver values in the ore into a shipping product.

Sampling and Analysis.—The entire shipment of 218 pounds was crushed to $\frac{1}{4}$ inch and cut twice; 53 pounds was crushed to 8 mesh and cut twice; 14 pounds was crushed to 20 mesh and the head sample cut from this latter quantity. Analysis showed it to contain:—

Gold.....	0.035 oz./ton	Copper.....	2.18 per cent
Silver.....	0.88 "	Iron.....	7.36 "
Arsenic.....	trace	Lead.....	none
Zinc.....	0.10 per cent	Insoluble.....	69.32 "

EXPERIMENTAL TESTS

A number of flotation tests were made using various reagents, and after grinding to various degrees of fineness. Grinding previous to flotation was done in porcelain pebble jars using grey iron balls and a pulp density of 50 per cent solids. Flotation tests were made in a Ruth machine. In test No. 7, a slightly different procedure was followed. In the previous tests 900 to 1,000 grammes of ore was used in each case, the result being that the pulp dilution in the cleaning of the rougher concentrate was too great, resulting in a high-grade middling product and a low recovery in each test except No. 6. In test No. 7, three lots of ore of 962, 964, and 979 grammes were floated separately, the concentrates being combined for cleaning. The results of the tests, and the reagents used, are given in the following table :—

Test No.	Mesh	Product	Weight per cent	Assays			Per cent of values		Reagent	Amount lb./ton	Added to
				Ag oz./ton	Au oz./ton	Cu per cent	Ag	Cu			
1	65	Concentrate.....	5.6	10.37	0.09	29.24	58.1	82.1	Lime.....	5.0	Ball mill.
		Middling.....	5.3	5.22	6.00	27.7	15.9	Xanthate.....	0.07	Ruth machine.
		Tailing.....	89.1	0.16	0.045	14.2	2.0	Pine oil No. 5.....	0.10	Ruth machine.
2	65	Concentrate.....	6.6	98.3	0.05	25.70	70.8	79.9	Water-gas tar, 50 per cent.....	0.40	Ball mill.
		Middling.....	9.7	2.41	3.63	25.6	16.6	Coal-tar creosote, 50 per cent.....	3.0	Ball mill.
		Tailing.....	83.7	0.04	0.09	3.6	3.5	Sodium carbonate.....	0.1	Ruth machine.
3	-48	Concentrate.....	4.1	6.90	31.56	32.4	58.7	Lime.....	5.0	Ball mill.
		Middling.....	6.6	6.76	12.17	51.2	36.4	Xanthate.....	0.07	Ruth machine.
		Tailing.....	89.3	0.16	0.12	16.4	4.9	Pine oil No. 5.....	0.1	Ruth machine.
4	65	Concentrate.....	4.9	7.20	0.10	30.18	40.1	67.6	Lime.....	2.5	Ball mill.
		Middling.....	6.2	5.92	9.87	41.7	27.5	QED.....	0.025	Ruth machine.
		Tailing.....	88.9	0.18	0.12	18.2	4.9	Pine oil No. 5.....	0.1	Ruth machine.
5	65	Concentrate.....	4.9	7.20	0.08	31.44	44.2	70.7	Lime.....	4.0	Ball mill.
		Middling.....	5.5	5.82	9.85	40.1	24.8	Water-gas tar, 50 per cent.....	0.4	Ball mill.
		Tailing.....	89.6	0.14	0.11	15.7	4.5	Coal-tar creosote, 50 per cent.....	0.1	Ruth machine.
6	65	Concentrate.....	13.1	6.70	16.48	88.3	94.1	Soda carbonate.....	3.0	Ball mill.
		Middling.....	6.7	1.02	0.84	6.8	2.4	Xanthate.....	0.5	Ruth machine.
		Tailing.....	80.2	0.06	0.10	4.9	3.5	Pine oil No. 5.....	0.10	Ruth machine.
7	Concentrate.....	8.6	8.06	24.68	85.9	94.0	Lime.....	5.0	Ball mill.
		Middling.....	3.1	1.10	0.85	3.9	1.1	Xanthate.....	0.07	Ruth machine.
		Tailing.....	88.3	0.10	0.125	10.2	4.9	Pine oil No. 5.....	0.10	Ruth machine.
									Xanthate for clean-up.....	0.01	Ruth machine.

SUMMARY

In tests Nos. 1 to 5 the recoveries are poor, though the grade of the copper concentrate is good. The middling in every case consisted of too large a quantity and was too high-grade to be retained in a mill circuit. In tests Nos. 6 and 7, the recoveries are good in the flotation concentrate, amounting on the average to 87 per cent of the silver and 94 per cent of the copper. The copper in test No. 7 concentrate being considerably in excess of that in No. 6. The middlings are more commensurate to those that would be obtained in actual mill practice.

Grinding to 65 mesh was found to be fine enough for good flotation results.

The gold values are concentrated with the silver and copper.

Potassium xanthate used in an alkaline circuit, preferably lime, gives good results, and it is coming into extensive use in flotation of copper ores. Its action is selective, exhibiting a preference for copper rather than iron. General Naval Stores pine oil No. 5 is an efficient frother.

Report No. 230

THE CONCENTRATION OF A COPPER-LEAD-ZINC ORE FROM ALBERT COUNTY, N.B.

C. S. Parsons

Shipment.—Two samples of copper-lead-zinc ore were submitted by John E. Teahan, Sr., Kerry P.O., N.B. The samples were obtained from a property near New Ireland, Alma parish, Albert county, N.B. Sample No. 1 was of 3 pounds; sample No. 2, of 116 pounds, was received at the Ore Testing Laboratories June 8, 1925, and was taken across a 30-foot face, 5 feet below the surface.

Purpose of Experimental Tests.—The samples were submitted for analysis and also to determine a method of concentration into commercial shipping products.

Character of the Ore.—The ore consisted of sulphides of iron, copper, lead, and zinc, finely crystalline and intimately associated. It also carried values in silver and gold.

Analysis of Samples.—Analysis of the two samples showed them to contain:—

	Sample No. 1	Sample No. 2
Copper.....	11.86 per cent	2.35 per cent
Arsenic.....	3.57 "	0.76 "
Lead.....	2.62 "	1.26 "
Zinc.....	7.44 "	14.27 "
Gold.....	0.44 oz./ton	0.03 oz./ton
Silver.....	14.94 "	3.91 "
Antimony.....	trace

EXPERIMENTAL TEST

The experimental work was conducted on sample No. 2. This sample taken across a 30-foot face, 5 feet below the surface, was considered to be truly representative of the class of ore to be expected from the development of the property. A selective flotation test was conducted making three products, namely a copper-lead concentrate, a zinc concentrate, and a tailing.

One thousand grammes of the ore, previously crushed to 14 mesh, was ground for 40 minutes in a ball mill. The following reagents were used to obtain a separation of the copper and lead from the zinc:—

Soda ash.....	10.0	lb./ton added to ball mill.
Thiocarbamide.....	0.25	" " "
Sodium cyanide.....	0.24	" " "
Pine oil.....	2	drops for lead-copper flotation.
Copper sulphate.....	1.5	lb./ton for zinc flotation.
Xanthate.....	0.3	" " "

The results obtained are given in the following table:—

Product	Weight		Analysis					Per cent of values				
	Grms.	%	Cu %	Pb %	Zn %	Au oz./ton	Ag oz./ton	Cu	Pb	Zn	Au	Ag
Copper-lead concentrate	169.7	17.0	12.32	6.62	13.86	0.08	14.76	90.3	93.3	16.3	41.6	70.0
Zinc concentrate.....	231.0	23.0	0.56	0.25	49.6	0.04	3.0	5.5	4.2	80.5	29.1	19.6
Tailing.....	600.0	60.0	0.16	0.05	0.7	0.015	0.56	4.1	2.5	3.0	28.3	9.5

SUMMARY AND CONCLUSIONS

The results show that the ore can be concentrated into two marketable products, namely a copper-lead product and a zinc product. The copper-lead concentrate obtained represented 17 per cent by weight of the ore; assayed 12.32 per cent copper, 6.62 per cent lead, 13.86 per cent zinc, 14.76 ounces silver per ton, and 0.08 ounce gold per ton. No attempt was made to reclean the concentrate to eliminate some of the zinc, which would be done in practice. This concentrate contained 90 per cent of the copper values in the ore, 93 per cent of the lead, 70 per cent of the silver, and 42 per cent of the gold.

The zinc concentrate obtained represented 23 per cent by weight of the ore; assayed 49.6 per cent zinc, with very small amounts of copper and lead, and is a good shipping product. This concentrate contained 80 per cent of the zinc values in the ore.

The tailing representing 60 per cent by weight of the ore carried very little values. It would seem, however, from the analysis and loss of values in the tailing that the gold values were to some extent associated with the iron sulphides, which reported in the tailing.

The test is an interesting one, showing what can be done by selective flotation on an intimate mixture of minerals in an ore.

Report No. 231

THE CONCENTRATION OF THE COPPER ORE ("C" ORE-BODY) OF THE
HORNE MINE, NORANDA MINES, LTD., ROUYN, QUE.

J. S. Godard

Shipment.—A shipment weighing 115 pounds was received June 15, 1925. It was designated chalcoppyrite, and was taken from "C" ore-body of the company's property at Rouyn, Que.

Characteristics of the Ore.—The ore consists of chalcoppyrite, pyrrhotite, and pyrite with which is associated some gold and silver. The gangue material is rhyolite.

Purpose of Experimental Tests.—The purpose of the tests was to concentrate the chalcoppyrite to the greatest possible extent commensurate with good recoveries of copper, gold, and silver.

Sampling and Analysis.—The entire sample was crushed to one-half inch and cut twice, then reduced to -4 mesh and cut once, further reduced to -14 mesh and cut twice. The last cut was ground to -150 mesh and a head sample cut out. Analysis was as follows:—

Gold.....	0.04 oz./ton	Copper.....	4.60 per cent
Silver.....	0.22 "	Zinc.....	none
Lead.....	none	Iron.....	19.83 "
Arsenic.....	trace	Insoluble.....	50.64 "

EXPERIMENTAL TESTS

Test No. 1

A sample of about 2,000 grammes of ore was cut from 24 pounds at -14 mesh and screened on Tyler standard screens. The products were assayed for gold and copper.

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
+ 20.....	10.8	0.05	4.03	9.6	9.3
- 20+ 28.....	14.0	0.04	4.01	10.0	12.0
- 28+ 35.....	16.5	0.05	4.07	14.8	14.4
- 35+ 48.....	11.9	0.04	4.08	8.6	10.4
- 48+ 65.....	9.2	0.05	4.18	8.2	8.2
- 65+100.....	7.5	0.07	4.95	9.4	8.0
-100+150.....	5.2	0.08	5.20	7.5	5.8
-150+200.....	4.8	0.08	5.57	6.8	5.7
-200.....	20.1	0.07	6.06	25.1	26.2

Test No. 2

Flotation, Tabling, and Screening.—One thousand grammes of ore at -14 mesh was ground wet for 20 minutes in a ball mill with 5 pounds of lime per ton, then floated in a Ruth machine with xanthate 0.07 pound

per ton, and No. 5 pine oil, 2 drops. The flotation tailings were tabled on a laboratory Wilfley table. The table tailings were screened.

Product	Weight per cent	Assays			Per cent of values	
		Au oz./ton	Cu per cent	Fe per cent	Au	Cu
Flotation concentrate.....	25.6	0.12	17.08	28.40	67.9	95.6
Table concentrate.....	1.9	0.08	0.10	44.40	3.3
Table middling.....	6.4	0.02	0.20	42.40	2.9	0.3
Table tailing.....	50.4	0.02	0.32	22.4	3.5
Slimes.....	15.7	0.01	0.18	3.5	0.6
- 65+100.....	7.9	0.015	0.62	6.3	15.1
-100+150.....	13.5	0.020	0.57	14.3	23.8
-150+200.....	13.4	0.015	0.41	10.6	17.0
-200.....	65.2	0.020	0.22	68.8	44.1
Average tailing.....	0.19	0.32

Tests Nos. 3 to 14—Flotation

The details, results of tests, and the flotation reagents used are given in the following table:—

Summary of Flotation Tests Nos. 3 to 14

Test No.	Mesh	Product	Weight per cent	Assays			Per cent of values		Reagent	Amount lb./ton	Added to	Remarks
				Au oz./ton	Cu %	Fe %	Au	Cu				
3	- 65	Concentrate.....	17.9	0.15	23.18	29.4	71.2	92.3	Lime.....	5.0	Ball mill....	No other reagents added for clean-up. Froth good.
		Middling.....	9.8	0.05	1.33		9.0	2.9	Xanthate.....	0.07	Ruth.	
		Tailing.....	72.3	0.015	0.30		19.8	4.8	Pine oil No. 5.....	0.05	Ruth.	
4	- 65	Concentrate.....	35.3	0.11	12.14	36.60	81.2	95.0	Soda carbonate.....	4.0	Ball mill....	Froth good. Concentrate, iron-looking in appearance.
		Middling.....	8.5	0.04	1.39		7.1	2.6	Xanthate.....	0.07	Ruth.	
		Tailing.....	56.2	0.01	0.19		11.7	2.4	Pine oil No. 5.....	0.03	Ruth.	
5	- 100	Concentrate.....	21.5	0.14	19.82	33.12	55.4	93.7	Lime.....	3.0	Ball mill....	Two lots ore, 1,000 grms. each, floated. Rougher concentrate cleaned together, tails combined. Froth dull—good.
		Middling.....	8.2	0.08	1.35	30.39	12.2	2.4	Xanthate.....	0.10	Ruth.	
		Tailing.....	70.3	0.025	0.25		32.4	3.9	Xanthate.....	0.01	Clean-up.	
6	- 100	Concentrate.....	15.0	0.19	27.62	29.38	39.5	91.5	Lime.....	4.0	Ball mill....	Froth dull in colour but good, 0.025 lb./ton, pine oil No. 5 added for clean-up.
		Middling.....	6.4	0.13	2.89	23.52	11.5	4.1	QED.....	0.05	Ruth.	
		Tailing.....	78.6	0.045	0.25		49.0	4.4	Pine oil No. 5.....	0.05	Ruth.	
7	- 100	Concentrate.....	22.9	0.16	17.20	35.66	47.1	86.0	Soda carbonate.....	4.0	Ball mill....	Froth bright—good, 0.025 lb./ton, pine oil No. 5 added for clean-up.
		Middling.....	15.0	0.15	2.47	27.78	29.0	8.1	QED.....	0.05	Ruth.	
		Tailing.....	62.1	0.03	0.44		23.9	5.9	Pine oil No. 5.....	0.05	Ruth.	
8	- 100	Concentrate.....	14.5	0.17	28.72	29.38	42.9	89.9	Lime.....	4.0	Ball mill....	Aero brand Ca(CN) ₂ used.
		Middling.....	8.2	0.07	3.00	21.29	9.9	5.3	Cyanide.....	0.3	Ball mill.	
		Tailing.....	77.3	0.035	0.29		47.2	4.8	Xanthate.....	0.07	Ruth.	
9	- 100	Concentrate.....	10.6	0.22	27.26	30.59	31.2	63.1	Soda carbonate.....	4.0	Ball mill....	Bakers C.P. NaCN. Bubbles small and numerous.
		Middling.....	13.4	0.14	7.37	24.54	25.1	21.6	Cyanide.....	0.3	Ball mill.	
		Tailing.....	76.0	0.043	0.92		43.7	15.3	Xanthate.....	0.07	Ruth.	
10	- 150	Concentrate.....	11.2	0.20	26.08	29.81	34.2	63.6	TT.....	1.0	Ball mill.	
		Middling.....	18.5	0.07	7.72	22.61	19.7	31.1	Cyanide.....	2.0	Ball mill.	
		Tailing.....	70.3	0.043	0.35		46.1	5.3				

11	-150	Concentrate..... Middling..... Tailing.....	12.5 6.4 81.1	0.16 0.23 0.05	31.24 3.78 0.32	29.96 27.62	26.6 19.5 53.9	88.6 Lime..... 5.5 Xanthate..... 5.9 Pine oil No. 5.....	5.0 0.07 0.025	Ball mill... Ruth.	Froth good—60 per cent tail- ings remained on 200 mesh.
12	-150	Concentrate..... Middling..... Tailing.....	13.4 6.6 80.0	0.19 0.12 0.035	30.20 7.12 0.35	30.16 22.57	41.5 12.9 45.6	84.4 Lime..... 9.8 Cyanide..... 5.8 Xanthate..... Pine oil No. 5.....	5.0 0.3 0.07 0.05	Ball mill... Ball mill... Ruth.	Froth good. Aero brand Ca (CN) ₂ .
13	-150	Concentrate..... Middling..... Tailing.....	12.5 7.4 80.1	0.16 0.12 0.03	30.98 4.80 0.41	30.68 21.34	37.8 16.8 45.4	85.0 Lime..... 7.8 Cyanide..... 7.2 Q&D..... Pine oil No. 5.....	4.0 0.3 0.05 0.05	Ball mill... Ball mill... Ruth.	Froth good.
14	-150	Concentrate..... Middling..... Tailing.....	9.5 10.0 80.5	0.16 0.12 0.043	28.58 10.37 1.00	29.14 24.62	24.6 19.4 56.0	58.6 TT..... 23.3 Cyanide..... 18.1 TT.....	1.0 2.0 0.03	Ball mill... Ball mill... Clean-up.	Bubbles small and test un- satisfactory.

SUMMARY AND CONCLUSIONS

1. In the first three flotation tests excellent copper recoveries were made. If the grade of the ore is taken into consideration, the gold recoveries obtained must also be considered as good. The loss of gold in the tailing varied between 20 and 30 cents, which gives a recovery of between 80 and 70 cents. In the latter tests the aim was to produce a high-grade copper concentrate and to determine the recovery of gold obtained with the different percentages of copper in the concentrate. It was found that the recovery of the gold dropped as the grade of the copper in the concentrate was increased. It is hoped that this information will be of assistance in determining the grade of copper concentrate from which the smelter will obtain the greatest profit.

2. From these results it is evident that the gold is not entirely associated with the chalcopyrite but must be either partly free or associated with the other sulphides. The results from tests on samples from "A" ore-body show that the pyrrhotite carries no gold and as pyrrhotite is easily dropped in any alkaline pulp, very little would float in any of the tests reported here. Therefore it follows that the gold is in all probability partly associated with chalcopyrite, partly with pyrite, and partly free, since from the results obtained in a screen analysis made on the flotation tailing from test No. 2, it will be observed that 68.8 per cent of the gold remaining in this tailing is in the -200-mesh product.

3. Crushing finer than 65 mesh did not improve the recovery of the gold or copper.

Report No. 232

THE CONCENTRATION OF A LEAD-ZINC ORE FROM THE KICKING HORSE MINE, FIELD, B.C.

C. S. Parsons

Shipment.—A shipment of 424 pounds of ore was received from the Kicking Horse mine, Field, B.C. It was submitted by the Pacific Mines, Ltd., 607-608 Pacific Building, Vancouver, B.C.

Characteristics of the Ore.—The ore consisted of galena and sphalerite carrying silver values. The sulphides were disseminated throughout the gangue, and required fine grinding in order to free them.

Purpose of Experimental Tests.—Experimental work was requested to determine the amenability of the ore to concentration and the methods of treatment most suitable to obtain marketable products with high recoveries of the contained values.

Analysis of Sample.—

Lead.....	7.16 per cent
Zinc.....	14.06 "
Silver.....	3.04 oz./ton
Gold.....	trace

CONCENTRATION TESTS

Each lot of ore used for a test was ground separately to 60 mesh. Reagents, as given below, were added and ground with the ore.

Test No. 1

Reagents—	
Added to ball mill—density of ball mill pulp.....	1 : 1
Soda ash.....	6.0 lb./ton
Low-grade calcium cyanide (Aero Brand).....	2.0 “
Thiocarbanilide.....	0.2 “
Added to cells—	
Lead cell—Cresylic acid.....	0.15 “
Zinc cell—Copper sulphate.....	1.0 “
Xanthate.....	0.4 “
Pine oil.....	1 drop

Test No. 2

Reagents—	
Added to ball mill—ball mill pulp density.....	1 : 1
Soda ash.....	4.0 lb./ton
Cyanide (Aero Brand).....	4.0 “
Thiocarbanilide.....	0.2 “
Added to cells—	
Lead cell—Pine oil.....	1 drop
Zinc cell—Copper sulphate.....	2.0 lb./ton
Xanthate.....	0.3 “
Pine oil.....	1 drop

NOTE.—Too much cyanide was used, and interfered with flotation of lead.

Test No. 3

Reagents—	
Added to ball mill—ball mill pulp density.....	1 : 1
Soda ash.....	4.0 lb./ton
Cyanide (Aero Brand).....	1.0 “
Thiocarbanilide.....	0.2 “
Zinc sulphate.....	1.0 “
Added to cells—	
Lead cell—Pine oil.....	1 drop
Zinc cell—Copper sulphate.....	2.0 lb./ton
Xanthate.....	0.3 “
Pine oil.....	1 drop

SUMMARY AND CONCLUSIONS

The use of zinc sulphate, together with cyanide, gives the best results. Attention is drawn to the fact that no attempt was made to reclean either the lead or zinc concentrate. By doing so the grade of both concentrates could be materially raised.

The separations were obtained without exercising special care in the manipulation of the machines, and there is no doubt that with ore similar in character to the sample received, equally as good, or better, results could be obtained in practice.

A recovery of 80 per cent of the lead in a concentrate assaying 50 per cent lead, and a recovery of 85 per cent of the zinc with a grade of 50 per cent, and a total recovery of the silver in the two concentrates of better than 90 per cent can be expected.

Results of Flotation Tests

Test No.	Product	Weight		Assay			Per cent of values		
		Grms.	Per cent	Pb Per cent	Zn Per cent	Ag oz./ton	Pb	Zn	Ag
1	Lead concentrate...	155.0	15.7	36.03	10.93	6.30	80.0	12.3	35.4
	Zinc concentrate....	250.7	25.3	4.25	45.33	6.25	15.2	82.5	56.9
	Tailing.....	584.0	59.0	0.57	1.22	0.36	4.8	5.2	7.7
2	Lead concentrate...	132.1	13.2	41.36	11.84	8.80	65.9	11.5	41.8
	Zinc concentrate....	244.4	24.5	9.67	40.27	5.52	28.5	72.3	48.8
	Tailing.....	622.2	62.3	0.75	3.54	0.43	5.6	16.2	9.5
3	Lead concentrate...	135.5	13.6	48.90	8.45	9.06	77.0	8.5	45.5
	Zinc concentrate....	243.7	24.5	6.25	46.95	5.30	17.7	84.7	47.9
	Tailing.....	615.5	61.9	0.75	1.49	0.29	5.3	6.8	6.6

Report No. 233

EXPERIMENTAL TESTS ON GOLD ORE FROM THE CONTACT MINES,
PAULSON, B.C.

J. S. Godard

Shipment.—A shipment of ore, 117 pounds, was received at the Ore Dressing and Metallurgical Laboratories, June 29, 1925, from the Contact Mines, Paulson, B.C. The shipment was made by Mr. P. H. Fraser, K.V. Mines, Ltd., Vancouver, B.C.

Characteristics of the Ore.—The ore consists of marcasite, with which is associated small quantities of the sulphides of lead, zinc, and copper, as well as gold and silver. The gangue material is siliceous. Platinum was reported to have been found in this ore, but analysis failed to disclose the presence of any of the metals of this group.

Sampling and Analysis.—The entire sample was crushed to $\frac{1}{4}$ inch and cut twice, then reduced to 14 mesh and cut once, then ground to -150 mesh before cutting the head sample. Analysis showed it to contain:—

Copper.....	0.05 per cent	Arsenic.....	trace
Lead.....	0.07 "	Silica.....	69.0 per cent
Zinc.....	0.12 "	Silver.....	0.80 oz./ton
Iron.....	10.95 "	Gold.....	0.48 "

Purpose of Experimental Tests.—Experimental tests were desired to determine methods for the recovery of the values. The proximity to a smelter makes a method of treatment that would concentrate the values into a shipping product for the smelter desirable.

EXPERIMENTAL TESTS

Test No. 1

Flotation and Tabling—

Two 1,000-gramme lots of ore were cut at -14 mesh, and ground to 100 mesh, then floated separately in a Ruth machine. The rougher concentrates from each lot were combined and cleaned. The tailings were combined and tabled on a small Wilfley table.

Product	Weight per cent	Assay Au oz./ton	Per cent of values
Flotation concentrate.....	15.1	2.32	72.9
Flotation middling.....	7.1	0.40	5.8
Table concentrate.....	1.1	1.86	4.2
Table tailing.....	47.1	0.11	10.8
Slimes.....	29.6	0.10	6.3

Analysis of the flotation concentrate gave—silver, 3.88 oz./ ton; lead, 0.46%; zinc, 0.46%; copper, 0.23%; arsenic trace; cobalt nil; nickel, nil; iron, 43.46%.

Flotation reagents—

Soda carbonate.....	5.0	lb./ton ball mill
Lot No. 1—Xanthate.....	0.6	" Ruth
TT mixture.....	0.10	" "
Pine oil No. 5.....	0.10	" "
Sulphuric acid.....	10.0	" "
Lot No. 2—Water-gas tar	1.0	" ball mill
Coal-tar creosote }		
Sulphuric acid.....	10.0	" Ruth
TT mixture.....	0.1	" "
Fumol No. 6.....	0.05	" "

No additional reagents used for cleaning concentrate.

Tests Nos. 2, 3, and 4—Cyanidation

Three cyanidation tests were made on 600-gramme lots of this ore. The table below gives the details.

No.	Mesh	Dilution	KCN per cent	Assay		Extraction per cent	Consumption lb./ton		Time hours
				Heads	Tails		KCN	CaO	
2.....	—100	1 : 2	0.10	0.48	0.02	95.8	2.0	4.0	62
3.....	—100	1 : 2	0.20	0.48	0.02	93.8	1.5	5.1	62
4.....	—200	1 : 3	0.15	0.48	0.02	95.8	2.2	4.3	62

All grinding done in pebble jar, discharge screened and oversize returned for further grinding.

Tests Nos. 5, 6, 7, 8, and 9—Flotation

Five flotation tests were made on lots of 1,000 grammes of ore at —14-mesh. Grinding was done in pebble jar.

Test No.	Product	Weight per cent	Assay		Per cent of values	
			Au oz./ton	Fe per cent	Au	Fe
5	Concentrate.....	20.4	1.04	37.33	49.3	79.4
	Middling.....	9.8	0.16	4.10	3.7	4.2
	Tailing.....	69.8	0.29	2.25	47.0	16.4
6	Concentrate.....	16.5	1.16	44.02	38.0	68.7
	Middling.....	7.2	2.20	20.14	31.5	13.7
	Tailing.....	76.3	0.20	2.44	30.5	17.6
7	Concentrate.....	18.2	1.56	44.17	59.9	76.2
	Middling.....	8.7	0.84	10.13	15.4	8.3
	Tailing.....	73.1	0.16	2.24	24.7	15.5
8	Concentrate.....	17.2	1.40	44.42	50.1	72.0
	Middling.....	9.4	1.15	14.24	22.4	12.6
	Tailing.....	73.4	0.18	2.22	27.5	15.4
9	Concentrate.....	13.2	1.24	43.76	34.1	57.8
	Middling.....	9.5	1.66	22.03	34.9	21.0
	Tailing.....	77.3	0.18	2.74	31.0	21.2

Reagents—

Test No.	Mesh	Reagents	Amount lb./ton	Added to	Remarks
5	100	Sodium sulphide.....	5.0	Ball mill	Concentrate cleaned in Janney machine without further reagents.
		Hardwood acid creosote			
		No. 27 FPL.....	0.3	"	
		Hercules tar oil.....	0.1	"	
		Pine oil No. 5.....	0.1	Ruth	
6	150	Barrett's No. 634.....	0.4	Ball mill	
		Pine oil No. 5.....	0.1	Ruth	
		Sulphuric acid.....	15.0	"	
7	65	Barrett's No. 634.....	0.4	Ball mill	No results from No. 634 in neutral pulp. Pine oil and sulphuric acid 3.5 lb./ton added for cleaning concentrate.
		Sulphuric acid.....	15.0	Ruth	
		Pine oil No. 350.....	0.45	"	
8	65	P. T. and T. No. 400....	0.28	Ball mill	Pine oil and sulphuric acid 3.5 lb./ton added for cleaning rougher concentrate.
		Pine oil No. 350.....	0.27	"	
		Sulphuric acid.....	18.0	Ruth	
9	150	P. T. and T. No. 400....	0.28	Ball mill	Pine oil and sulphuric acid 3.5 lb./ton added for cleaning rougher concentrate.
		Pine oil No. 350.....	0.27	"	
		Sulphuric acid.....	18.0	Ruth	

Test No. 10

Amalgamation and Tabling.—One thousand grammes ore —14 mesh was ground wet to 48 mesh and amalgamated. The amalgamation tailing was tabled.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
Concentrate.....	8.2	0.60	44.57	23.2	33.3
Middling.....	4.0	0.48	41.82	9.0	15.2
Tailing.....	63.2	0.18	6.73	54.0	38.8
Slimes.....	24.6	0.12	5.69	13.8	12.7

Head sample—gold.....	0.48	oz./ton
After amalgamation.....	0.211	"
Gold amalgamated.....	56.0	per cent
Recovery—		
By amalgamation.....	56.0	"
In table concentrate.....	10.3	"
Total recovery.....	66.3	"

Test No. 11

Amalgamation at 48 Mesh.—Results of screen test on amalgamation tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
+ 65.....	1.2	0.24	86.2	1.4	1.0
— 65+100.....	7.4	0.32	13.08	11.7	9.1
—100+150.....	13.3	0.28	13.62	18.3	17.1
—150+200.....	13.2	0.22	12.09	14.3	15.1
—200.....	64.9	0.17	9.44	54.3	57.7

Head sample—gold.....	0.48	oz./ton
After amalgamation.....	0.203	"
Gold amalgamated.....	57.7	per cent

Test No. 12

Amalgamation and Flotation.—One thousand grammes ore at -14 mesh, ground wet to 48 mesh and amalgamated. The amalgamation tailing was floated and the flotation tailing given a screen analysis.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
Concentrate.....	14.1	0.60	43.30	38.3	64.1
Middling.....	8.4	0.43	19.18	16.2	16.9
Tailing.....	77.5	0.13	2.34	45.5	19.0
Screen test—					
+ 65.....	4.9	0.14	3.74	5.2	7.8
— 65+100.....	7.2	0.15	3.00	8.1	9.2
— 100+150.....	12.7	0.16	2.55	15.2	13.8
— 150+200.....	8.5	0.18	2.21	11.5	8.0
— 200.....	66.7	0.12	2.15	60.0	61.2

Reagents—

P. T. and T. No. 400.....	0.28 lb./ton for rougher concentrate
Pine oil No. 350.....	0.45 “ “ “
Sulphuric acid.....	20.0 “ “ “
TT mixture.....	0.05 “ “ “
Sulphuric acid.....	6.0 “ for cleaning concentrate
Pine oil No. 350.....	0.27 “ “ “
Pine oil No. 5.....	0.05 “ “ “

Recovery—

By amalgamation.....	53.7 per cent
In flotation concentrate.....	17.8 “
Total.....	71.5 “

Test No. 13

Amalgamation and Flotation.—The amalgamation tailings were re-ground for flotation. Flotation heads: gold, 0.21 ounce per ton; iron, 10.22 per cent.

—	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Fe per cent	Au	Fe
Concentrate.....	13.5	0.64	43.10	40.2	57.0
Middling.....	10.5	0.42	15.05	20.5	15.4
Tailing.....	76.0	0.11	3.72	39.2	27.6

Reagents—

P. T. and T. No. 400.....	0.28 lb./ton added to ball mill
Pine oil No. 350.....	0.45 “ “ “
Sulphuric acid.....	20.00 “ “ Ruth machine
Pine oil No. 350.....	0.16 “ for cleaning concentrate
Sulphuric acid.....	6.0 “ “ “

Recovery—

By amalgamation.....	56.2 per cent
In flotation concentrate.....	18.1 “
Total.....	74.3 “

Test No. 14

Amalgamation and Flotation.—The amalgamation tailing was re-ground for flotation. Flotation heads: gold, 0.17 ounce per ton; silver, 0.77 ounce per ton.

Product	Weight per cent	Assay		Per cent of values	
		Au oz./ton	Ag oz./ton	Au	Ag
Concentrate.....	28.0	0.46	2.24	75.0	81.3
Tailing.....	72.0	0.06	0.20	25.0	18.7

Recovery—

By amalgamation.....	gold	64.2 per cent	silver	14.3 per cent
In flotation concentrate.....	"	26.8 "	"	69.9 "
Total.....	"	91.0 "	"	84.2 "

Reagents—

Heavy hardwood oil, S. T. and C. Co.....	0.56 lb./ton to ball mill
Pine oil No. 350.....	0.27 " "
Sulphuric acid.....	15.0 " to Ruth
Pine oil No. 350.....	0.27 " "

Tests Nos. 15, 16, and 17—Flotation at 150 Mesh—

Test No.	Product	Weight per cent	Assay		Per cent of values	
			Au oz./ton	Ag oz./ton	Au	Ag
15	Concentrate.....	30.0	1.18	2.50	80.0	81.4
	Tailing.....	69.6	0.13	0.25	20.0	18.6
16	Concentrate.....	30.1	1.60	2.70	93.2	89.2
	Tailing.....	69.9	0.05	0.14	6.8	10.8
17	Concentrate.....	24.3	1.50	2.92	85.9	82.5
	Tailing.....	75.7	0.08	0.20	14.1	17.5

Test No.	Reagents	Quantity lb./ton	Added to
15	Sodium sulphide.....	5.0	Ball mill.
	Dupont's flotation oil.....	0.40	"
	Pine oil No. 350.....	0.16	"
	Sulphuric acid.....	18.0	Ruth machine.
	Pine oil No. 5.....	0.02	"
	Fumol No. 6.....	0.02	"
16	Soda carbonate.....	10.0	Ball mill.
	Xanthate.....	0.80	Ruth machine.
	Pine oil No. 5.....	0.06	"
17	Barrett's No. 4.....	0.40	Ball mill.
	Pine oil No. 350.....	0.16	"
	Pine oil No. 350.....	0.16	Ruth machine.
	Pine oil No. 5.....	0.08	"
	Sulphuric acid.....	16.00	"

SUMMARY

Amalgamation.—Recoveries of the gold by amalgamation at 48 mesh varied from 53 to 64 per cent, with an average of 56 per cent; 14.3 per cent of the silver was amalgamated in test No. 14. The mercury was in good condition after each test.

Cyanidation.—Three small-scale cyanidation tests were made. All three showed good extraction of the gold with moderate cyanide consumption. Using a dilution of 1 : 2 and cyanide of 0.10 per cent on ore ground to -100 mesh was equally as effective as more dilute pulp and stronger cyanide on -200-mesh material.

Tabling.—Tabling the amalgamation tailings did not prove very satisfactory, as considerable gold values remained in the tailings and slimes. Large-scale operations on a classified feed would greatly improve the recovery over that obtained in small-scale work.

Flotation.—A great variety of results were obtained by flotation. The results in the first tests made were disappointing, but this was largely due to part oxidation of the pyrites. Fresh samples of ore were crushed for tests Nos. 14, 15, 16, and 17, with greatly improved recoveries. Test No. 16, using a soda pulp and xanthate, was the most satisfactory. Should an acid pulp be used, a considerable reduction in the amount of the sulphuric acid used could be made on mill-scale operations and freshly broken ore. In the latter tests the froth was good, though the sulphides were slightly sluggish.

CONCLUSIONS

Amalgamation alone is not sufficient to yield good recoveries, but it could be supplemented by tabling the classified amalgamation tailing, or by flotation. Results show that amalgamation should be considered in the treatment of this ore.

Flotation.—Good recoveries can be obtained by flotation. The best results were obtained at 150 mesh. It is a question whether the additional recoveries made at 150 mesh over those at 65 mesh would warrant the extra grinding costs. Flotation could be adopted either alone or after amalgamation.

Cyanidation.—Good recoveries of the values were obtained by cyanidation. However, before adopting this method of treatment, due consideration should be given to the possibility of the ore becoming more refractory, in which case the results by this method would not be so satisfactory.

Report No. 234

THE CONCENTRATION OF A ZINC ORE FROM RENFREW, ONT.

J. S. Godard

Shipments.—Two shipments of zinc ore designated T-1 and T-2 were received at the Ore Dressing and Metallurgical Laboratories during September, 1925, from Renfrew, Ont. Sample T-1 weighed 85.5 pounds and sample T-2, 96.5 pounds. The shipment was made by Messrs. Alderson and MacKay, Montreal.

Characteristics of the Ore.—The ore is a medium-coloured zinc blende in an altered calcareous gangue. Considerable mica is present.

Purpose of Experimental Tests.—The object in conducting the tests was to determine the amenability of the ore to concentration, the grade of the zinc product, and the recovery of the zinc values that could be expected.

Analysis.—The two samples were found to contain on analysis:—

Sample	Lead per cent	Zinc per cent	Silver oz./ton	Gold oz./ton
T-1.....	nil	11.75	trace	nil
T-2.....	trace	18.42	trace	nil

EXPERIMENTAL TESTS

Flotation tests were made on each sample, using 1,000 grammes of ore, and a laboratory Ruth flotation machine. Reagents used:—

Soda carbonate.....	5.0 lb./ton added to ball mill
Xanthate.....	0.6 " " Ruth
Pine oil No. 5.....	0.05 " " "

The results of the tests are given in the following tables:—

Sample No.	Product	Weight per cent	Assay			Per cent of zinc values
			Zn per cent	Ag oz./ton	Fe per cent	
T-1	Concentrate.....	24.2	47.69	0.04	94.6
	Middling.....	9.3	4.37	3.3
	Tailing.....	66.5	0.39	2.1
T-2	Concentrate.....	30.3	57.63	trace	3.48	93.1
	Middling.....	12.9	8.13	5.5
	Tailing.....	56.8	0.47	1.4

Screen tests on the tailings show the following grinding results:—

Mesh	Weight per cent		Cumulative per cent	
	T-1	T-2	T-1	T-2
+ 48.....	1.3	1.3
+ 65.....	2.9	0.6	4.2	0.6
+ 100.....	14.1	13.3	18.3	13.9
+ 150.....	13.2	11.8	31.5	25.7
+ 200.....	13.7	13.5	45.2	39.2
- 200.....	54.8	60.8	100.0	100.0

SUMMARY AND CONCLUSIONS

Straight flotation produced very good results on ore typified by samples T-1 and T-2. A good grade of concentrate, with high recoveries, is easily made.

Report No. 235

THE CONCENTRATION OF A LEAD ORE FROM THE FRONTENAC MINE,
PERTH ROAD, ONT.

J. S. Godard

Shipment.—A shipment of 56 pounds consisting of 5 different samples was received at the Ore Testing Laboratories, September 30, 1925, from the Frontenac mine, Perth Road, Ont. The ore was submitted by Mr. J. M. Forbes, Ottawa.

Characteristics of the Ore.—The ore consisted of galena associated with a small quantity of zinc blende, in a calcite gangue.

Purpose of Shipment.—All samples were to be analysed for lead and zinc, and samples Nos. 3, 4, and 6 were to be given a table test.

Sampling and Analysis:—

Sample No.	Weight lbs.	Lead per cent	Zinc per cent
2.....	4.0	7.92	0.39
3.....	14.5	3.42	0.48
4.....	16.0	2.38	0.50
5.....	5.0	3.89	0.32
6.....	16.0	3.37	0.10

EXPERIMENTAL TESTS

One thousand grammes of each of the three samples was tabled separately on a small Wilfley table. Tabling was done on material crushed to pass a 20-mesh Tyler screen.

Results

Sample No.	Product	Weight per cent	Analyses		Per cent of values Pb
			Pb per cent	Zn per cent	
3	Concentrate.....	3.5	65.14	2.94	77.5
	Middling.....	34.7	0.52	0.51	6.2
	Tailing.....	53.7	0.74	13.5
	Slimes.....	8.1	1.04	2.8
4	Concentrate.....	2.5	67.52	2.96	76.1
	Middling.....	18.0	1.36	1.05	11.0
	Tailing.....	70.6	0.35	11.1
	Slimes.....	8.9	0.45	1.8
6	Concentrate.....	2.9	74.94	0.44	67.6
	Middling.....	8.3	2.62	0.07	6.8
	Tailing.....	79.4	0.97	23.9
	Slimes.....	9.4	0.59	1.7

SUMMARY AND CONCLUSIONS

The table concentrates contained from 65 to 75 per cent lead, with recoveries (not including the middlings) of from 68 to 76 per cent. Although these recoveries seem low, it should be borne in mind that the feed was —20 mesh, dry, and unsized, and that the operations were conducted on a small table. In actual practice, the coarse material would be jigged and the finer tabled, which, with the re-running of the middling should recover from 85 to 90 per cent of the lead values in the ore. The

slimes from jig and table concentration of the sized and classified feed could be thickened and the values recovered in a small flotation unit. If sufficient zinc were present in the ore to warrant it, a zinc middling product could be cut from the tables, reground, and floated into a marketable zinc product.

Report No. 236

THE CONCENTRATION OF A COPPER-ZINC ORE FROM THE AMULET MINE, ROUYN, QUE.

C. S. Parsons

Shipment.—A shipment of 80 pounds of copper-zinc ore was received from the Amulet mine, Rouyn mining district, Quebec, October 13, 1925. The sample was marked "Composite No. 1a".

Analysis and Characteristics.—The sample contained copper and zinc with gold and silver values, associated with iron sulphides. The copper and zinc sulphides were finely disseminated, necessitating fine grinding to free them. Analysis of the sample showed it to contain the following:—

Copper.....	4.15 per cent
Zinc.....	10.00 "
Gold.....	0.14 oz./ton
Silver.....	3.02 "

Purpose of Experimental Tests.—The following tests were made with the object of separating the copper and zinc sulphides, and determining where the gold and silver values would report.

EXPERIMENTAL TESTS

A series of four selective flotation tests was run, different reagents being used in each test. The ore was ground to pass approximately 65 mesh. Two amalgamation tests were made for the recovery of the gold. One table test was made for recovery of the gold remaining in the flotation tailing.

Results of Flotation Tests:—

Test No.	Product	Weight per cent	Analysis				Per cent of values			
			Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Cu	Zn	Au	Ag
1	Copper concentrate..	22.0	17.20	11.33	0.24	7.80	90.1	24.5	39.4	59.2
	Zinc concentrate....	14.7	1.49	47.55	0.22	3.38	5.2	69.0	24.0	17.2
	Zinc middling.....	13.8	0.79	4.09	0.32	3.80	2.6	5.5	32.9	18.6
	Tailing.....	49.5	0.18	0.20	0.01	0.33	2.1	1.0	3.7	5.6
2	Copper concentrate..	15.6	21.78	11.13	0.20	9.0	82.6	17.1	21.7	46.9
	Copper middling....	4.6	3.71	17.27	0.18	5.38	4.2	7.9	5.8	8.3
	Zinc concentrate....	12.2	1.41	51.61	0.04	1.56	4.2	62.2	3.4	6.3
	Zinc middling.....	6.8	2.05	13.02	0.20	4.00	3.4	8.8	0.5	9.2
	Tailing.....	60.8	0.38	0.66	0.14	1.44	5.6	4.0	59.6	29.3
3	Copper concentrate..	21.5	17.62	9.91	0.46	10.66	90.7	21.0	71.6	75.1
	Zinc concentrate....	14.3	0.74	41.5	0.02	0.98	2.5	58.1	2.1	4.4
	Zinc middling.....	9.3	1.06	16.15	0.035	1.64	2.3	14.7	2.4	5.0
	Tailing.....	54.9	0.34	1.15	0.06	0.86	4.5	6.2	23.9	15.5
4	Copper concentrate..	17.3	20.66	6.91	0.56	11.96	85.5	11.7	50.7	65.9
	Zinc concentrate....	14.6	1.96	48.57	0.32	1.39	6.9	70.0	13.6	6.6
	Zinc middling.....	8.1	1.30	11.37	0.30	3.80	2.6	9.1	23.1	9.9
	Tailing.....	61.0	0.35	1.56	0.04	0.93	5.0	9.2	12.6	17.6

Results of Amalgamation Tests:—

Test made on head sample—

Heads, gold.....	0.14	oz./ton
Tailing, gold.....	0.11	"
Recovery of gold.....	21.4	per cent

Test made on flotation tailing from Test No. 2—

Heads, gold.....	0.14	oz./ton
Tailing, gold.....	0.015	"
Recovery of gold.....	89.3	per cent

Results of Table Test:—

Test made on flotation tailing from Test No. 4—

Product—	Gold oz./ton	Silver oz./ton
Heads.....	0.04	0.92
Table concentrate.....	0.79	2.42
Table tailing.....	0.02	0.96
Recovery of gold.....	68.8 per cent	
Table concentrate represented.....	3.7 per cent of feed	

DISCUSSION

The tests indicate that a remarkably good separation of the copper and zinc can be obtained, the copper concentrate of test No. 4 having the lowest content of zinc.

With respect to the gold and silver values, the erratic way in which they report in the different products indicates that a part of the gold is present in the ore in the free state. The writer is not inclined to view this erratic behaviour of the gold as due to the different reagents used in the various tests. The gold remaining in the flotation tailing can be reduced to 0.015 ounce per ton by amalgamation, as indicated by the test on the tailing from test No. 2, which gave a recovery of 89 per cent of the remaining gold. The amalgamation of the ore before flotation was not so successful, as a recovery of only 21 per cent was obtained.

Too much weight should not be placed on these amalgamation tests, as it will be necessary to have large-scale tests made to determine the most efficient place to introduce amalgamation, especially taking into consideration the amount of gold reporting in the zinc concentrate, in which product it is not desired. It must also be determined where the gold which is in the middling product will report in actual operation when this middling is returned continuously to the circuit. It is possible that it would be best to amalgamate after copper flotation and before zinc flotation. It is practically impossible to determine these points from small-scale laboratory work.

Referring to the table test made on the flotation tailing from test No. 4: A good recovery of the gold was obtained in a concentrate representing 3.7 per cent of the weight of the flotation tailing and assaying 0.8 ounce per ton, with a tailing assaying between 0.015 and 0.02 ounce per ton.

The presence of free gold in the sample may be due to the samples from which this composite sample was made up being taken close to the surface and becoming contaminated by free gold from the gossan capping, and that the gold will not occur in this state in the ore below surface oxidation. If this supposition is correct, amalgamation to recover the gold at some point in the process could be dispensed with.

SUMMARY

The results of the tests show that the ore is amenable to concentration by selective flotation of the copper from the zinc. High-grade copper concentrate can be obtained with recoveries of 90 per cent of the copper values. This concentrate is fairly high in zinc and a good deal of experimental work could be done to obtain a better selective action between the copper and the zinc so as to lower the zinc content. It would seem that this could be more readily done by re-treatment of the copper concentrates. High-grade zinc concentrate, containing 50 per cent zinc can be obtained and the recovery of zinc in this concentrate should be around 75 per cent.

With respect to the gold values, if the sample submitted is representative of the ore in depth, by supplementing flotation with either amalgamation or tabling, the amount of gold lost in the final tailing can be reduced to about 30 cents per ton. Part of the gold recovered reports in the zinc concentrate. The smelters allow very little for gold in a zinc concentrate, as it is difficult to recover. One of the aims of any further test work should be to prevent the gold and silver values reporting with the zinc concentrate.

Report No. 237

THE CONCENTRATION OF A ZINC-SILVER ORE FROM THE WONDERFUL MINE, SANDON, B.C.

J. S. Godard

Shipment.—A shipment of 235 pounds of a zinc-silver ore from the Wonderful mine, Sandon, B.C., was received at the Ore Dressing and Metallurgical Laboratories October 22, 1925. It was designated as sample of feed to the Alamo concentrator, operated by the Cunningham Mines Ltd. at Alamo.

Characteristics of the Ore.—The ore was tentatively classed as lead-zinc-silver, but on analysis was found to contain only 0.35 per cent lead. The principal gangue mineral is silica, but some graphite as graphite schist, some siderite, and a small quantity of a manganese mineral, presumably rhodochrosite, are present. Silver amounting to about 8 ounces per ton is associated with the sulphides, principally the zinc.

Sampling and Analysis.—The ore as received, $\frac{1}{2}$ -inch size, was thoroughly mixed and cut twice in a Jores sampler. One cut was ground to -14 mesh and cut twice, then further reduced to -48 mesh before cutting the head sample. Analysis showed the ore to contain

Lead.....	0.35 per cent	Zinc.....	9.52 per cent
Silver.....	7.80 oz./ton	Insoluble.....	61.70 "
Sulphur.....	7.09 per cent	Manganese, determined qualitatively	

Purpose of Experimental Tests.—The flow-sheet at the Alamo concentrator consisted of grinding, screening on a Callow belt screen, tabling out the lead, dewatering the table tailing and slimes, and flotation of the slimes for recovery of the zinc. The loss of silver was high, and the recoveries of both lead and zinc were poor. Selective flotation was considered as a means of effecting a saving of the silver and increasing the recoveries

and grades of concentrates in the case of both the lead and zinc. The possibility of making a zinc concentrate containing the silver and disregarding the small quantity of lead present was also considered. With the above two purposes in view, tests as tabulated below were undertaken:

Test No.	Product	Weight per cent	Assays			Per cent of values		
			Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
1	Lead concentrate.....	10.4	4.01	2.95	13.56	99	3.1	18.5
	Zinc concentrate.....	21.4	tr.	43.18	28.00	94.6	78.5
	Tailing.....	68.2	tr.	0.33	0.33	2.3	3.0
2	Lead concentrate.....	11.6	3.80	2.81	13.50	99	3.3	20.0
	Zinc concentrate.....	20.8	tr.	44.50	29.00	95.3	77.0
	Tailing.....	67.6	tr.	0.21	0.36	1.4	3.0

The tailings from tests Nos. 1 and 2 were screened:—

Test No.	Mesh	Weight per cent	Cumulative per cent
1.....	+ 65	0.5	0.5
	+100	9.4	9.9
	+150	11.6	21.5
	+200	13.0	34.5
	—200	65.5	100.0
2.....	+ 65	0.1	0.1
	+100	1.9	2.0
	+150	6.8	8.8
	+200	10.7	19.5
	—200	80.5	100.0

Reagents Used for Tests Nos. 1 and 2:—

Lead reagents—			
Soda carbonate.....	5.0 lb./ton	added to ball mill	
Thiocarbamilide.....	0.2	"	"
Cyanide.....	0.5	"	"
Pine oil No. 5.....	0.04	"	Ruth
Zinc reagents—			
Copper sulphate.....	2.0	"	"
Xanthate.....	0.5	"	"
Pine oil No. 5.....	0.04	"	"

Summary.—There is very little difference in results between these tests. Evidently, grinding as in No. 1 is sufficiently fine to make the separation.

Test No. 3

Selective Flotation at 65 Mesh—

Six lots of ore of 1,000 grammes each at —14 mesh were ground and floated separately in a Ruth machine. The lead concentrates were combined and cleaned. The zinc concentrates were similarly treated. All the tailings were combined.

Product	Weight per cent	Assays			Per cent of values		
		Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
Lead concentrate.....	4.5	8.84	1.41	22.36	92.2	0.7	13.4
Lead middling.....	5.9	0.19	5.27	5.08	2.5	3.2	4.0
Zinc concentrate.....	16.8	0.14	53.11	33.60	5.3	92.9	75.3
Zinc middling.....	7.2	tr.	2.91	3.54	2.2	3.4
Tailing.....	65.6	tr.	0.15	0.44	1.0	3.9

Reagents: Same as in tests Nos. 1 and 2.

Summary.—Good recoveries were made of the lead, zinc, and silver values. The amount of lead concentrate appeared to be much more than the analysis showed, due to the presence of graphite which floated with the lead. The carbon in the lead concentrate amounted to 5.64 per cent.

Test No. 4

Flotation at 65 Mesh—

The purpose of the test was to concentrate the lead, no attempt being made to concentrate the zinc. Eight lots of ore of 1,000 grammes each were ground separately to 65 mesh and floated separately in a Ruth machine. The cleaning of the concentrate was done as follows. Four rougher concentrates were combined and cleaned making a concentrate, then the remaining four rougher concentrates were treated similarly. Both cleaner concentrates were combined and recleaned, and the concentrates from this recleaned, making in all three middling products.

	Weight per cent	Assays			Per cent of values		
		Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
Lead concentrate.....	4.9	7.98	1.05	21.52	95.4	0.5	14.5
Middling No. 1.....	6.4	0.11	5.67	4.54	1.7	3.9	4.0
Middling No. 2.....	1.3	0.25	3.01	3.42	0.7	0.4	0.6
Middling No. 3.....	0.9	0.98	2.36	4.68	2.2	0.2	0.6
Tailing.....	86.5	tr.	10.24	6.74	95.0	80.3

Reagents—

Soda carbonate.....	5.0	lb./ton added to ball mill
Thiocarbamilide.....	0.20	" " "
Cyanide.....	0.50	" " "
Pine oil No. 5.....	0.04	" " Ruth

Summary.—It seems impossible to raise the lead content of the concentrate above 9 per cent (test No. 3) by flotation. Graphite again appeared in the lead concentrates. It is darker than the galena when seen in the cells and forms a very thin coating over the bubbles. The carbon in the lead concentrate amounted to 6.56 per cent.

Tests Nos. 5 and 6

Flotation at 65 mesh—

In tests Nos. 5 and 6, the lead was disregarded and the ore treated as a zinc-silver ore.

Test No.	Product	Weight per cent	Assay		Per cent of values	
			Zn per cent	Ag oz./ton	Zn	Ag
5	Zinc concentrate.....	20.0	45.68	32.94	96.8	91.0
	Zinc middling.....	10.4	1.30	1.94	1.4	2.8
	Tailing.....	69.6	0.25	0.64	1.8	6.2
6	Zinc concentrate.....	16.6	52.71	37.90	92.5	87.6
	Zinc middling.....	8.6	5.32	6.18	4.8	7.4
	Tailing.....	74.8	0.35	0.48	2.7	5.0

Reagents Used—

Test No. 5—Soda carbonate.....	5.0	lb./ton added to ball mill	
Copper sulphate.....	2.0	"	Ruth
Xanthate.....	0.4	"	"
Pine oil No. 5.....	0.08	"	"
Test No. 6—Lime.....	5.0	"	ball mill
Copper sulphate.....	2.0	"	Ruth
Xanthate.....	0.4	"	"
Pine oil No. 5.....	0.08	"	"

Summary.—As far as recoveries are concerned there is little to choose between soda carbonate and lime as a means of producing alkalinity. Slightly better zinc recoveries were obtained with soda carbonate, although the silver results were slightly better when lime was used. The froth in the Ruth machine, using soda carbonate is more active, but if a pneumatic type of cell were used, lime could possibly be utilized at a lower cost.

SUMMARY AND CONCLUSIONS

Selective Flotation.—Good recoveries of all three metals can be made by selective flotation. There was not sufficient lead values in the ore to make a high-grade lead concentrate.

Straight Flotation.—When the lead content is disregarded and the ore treated as a zinc-silver ore, good recoveries of both zinc and silver are obtained. The concentrate is of good grade, averaging 50 per cent zinc and 35 ounces silver per ton, with recoveries of 95 per cent of the zinc and 90 per cent of the silver values. No difficulty should be experienced in duplicating the results of these tests in mill-scale operations.

The reagents used were found entirely satisfactory so no others were tried. If the ore be considered as a straight zinc-silver proposition the alkaline reagent should be added to the ball mill in the closed circuit. The best results in using xanthate should be obtained by adding it in either solid form or in solution to a pump, or some other mixing device, previous to entering the flotation cell. Pine oil if used as a frother should be injected just following the xanthate.

III

REPORTS OF INVESTIGATIONS: ELECTROCHEMICAL AND
HYDROMETALLURGICAL LABORATORIESHYDROMETALLURGICAL TREATMENT OF IRON SULPHIDE ORES FOR THE
PRODUCTION OF ELECTROLYTIC IRON, WITH THE RECOVERY
OF SULPHUR AND OTHER METALS AS BY-PRODUCTS

R. J. Traill and W. R. McClelland

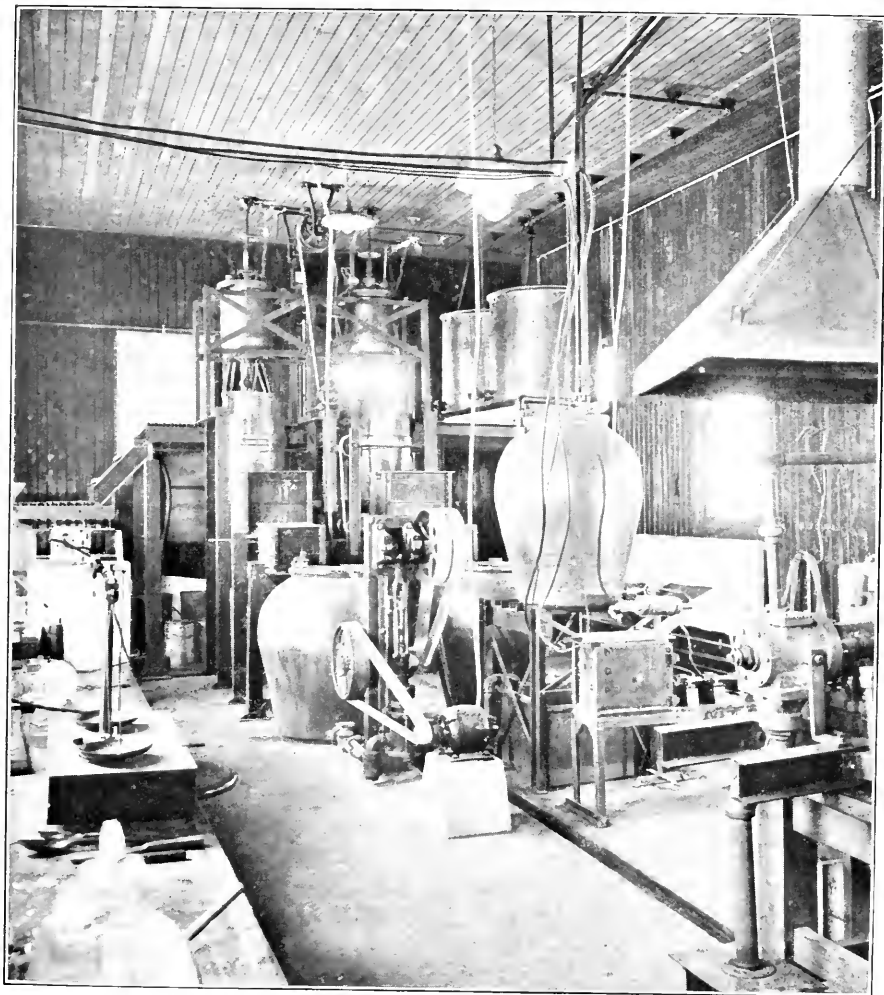
INTRODUCTORY

The leaching experiments on pyrrhotite and roasted pyrite were continued along the same lines as described in the report for 1924, and were carried forward to the purification of the electrolyte and the electro-deposition of iron. Previous experiments showed quite definitely the importance of having a pure electrolyte to obtain good deposits of iron. Lead appears to be the most harmful impurity, and its almost complete elimination seems necessary, and although copper and zinc are less harmful, they should be eliminated as completely as possible for best results. Tests for the purpose of eliminating these interfering elements were made and will be briefly described.

Some progress has been made in electro-deposition; good deposits have been obtained, and cell operation has been fairly well standardized. The essential factors governing iron electro-deposition would seem to be purity, high temperature, and acidity of electrolyte. The purity is the most difficult to control and although this has been partly accomplished, it has by no means been completely solved and further work is contemplated along this line. With satisfactory electrolyte, proper design of cell, and careful control of operating conditions, it appears evident that current efficiencies averaging 95 per cent are quite possible.

LEACHING OF PYRRHOTITE

Full details relative to leaching pyrrhotite are given in the report of 1924. The method described therein has been followed throughout these tests, and may be summarized here as treating the finely ground (–200 mesh) ore in stoneware agitation jars with a ferric-ferrous chloride solution at a temperature of 95°C. Investigation showed, however, that freshly ground ore reacted much more rapidly than ore which had been ground and left standing for some time. Apparently surface oxidation occurs when the crushed ore is allowed to stand, and this exerts a retarding influence on the leaching of the ore. With oxidized ore the time of leaching is from 50 to 100 per cent greater than with fresh ore and there is a tendency toward greater hydrolysis occurring in the liquor.



Equipment of the electrochemical laboratory for electrolytic iron investigations.

CONVERSION OF PYRITE TO PYRRHOTITE

Pyrite is not readily soluble in ferric chloride and requires a preliminary treatment which consists of roasting in a non-oxidizing atmosphere whereby one atom of recoverable sulphur is distilled off and an artificial pyrrhotite is formed. The furnace and operation are described in the report of 1924. Numerous tests were carried out on Eustis pyrite (Eustis, Quebec) ore and fairly consistent results obtained. The recoveries of sulphur were not so high as desired, but increased recoveries should not be a problem of any great difficulty. The product resulting from this roasting process conforms closely to the formula Fe_3S_3 , and it is a much more soluble material than natural pyrrhotite. With the furnace as designed it was necessary to make three passes of the ore to reach approximate total elimination of the one atom of sulphur. The following are typical examples of the results:—

(A) Charge 30 pounds, —40 mesh.....	41.2 per cent iron.
Time of each pass.....	1½ to 2 hours.
Analysis after 1st pass.....	21.43 per cent chloride soluble iron.
2nd pass.....	40.14 “ “ “
3rd pass.....	49.10 “ “ “
	1.83 “ copper.
	0.70 “ lead.
Sulphur recovered.....	3.0 pounds.
(B) Charge 30 pounds, —40 mesh.....	41.2 per cent iron.
Time of each pass.....	about 2 hours.
Analysis after 1st pass.....	20.42 per cent chloride soluble iron.
2nd pass.....	38.45 “ “ “
3rd pass.....	48.9 “ “ “
Sulphur recovered.....	3.0 pounds.

The ore used in these tests had oxidized to some extent, and the oxide apparently picked up free sulphur forming an iron sulphide. Further sulphur was lost by condensing in cooler parts of the furnace and through initial conversion with the oxygen in the furnace to sulphur dioxide. The sulphur recovery is less than 50 per cent.

LEACHING ARTIFICIAL PYRRHOTITE

Leaching tests on roasted pyrite or artificial pyrrhotite have been described in the report for 1924. The rate of solubility is about three times that of natural pyrrhotite, probably on account of its spongy condition when obtained from the furnace. The effect of surface oxidation is also noticeable in this material and long standing of the product results in slow leaching rate with attendant hydrolysis. The separation of the residue and leach liquor by filtration is a much simpler problem in this case than it is with natural pyrrhotite.

METAL IMPURITIES AFFECTING DEPOSITION

Most pyrrhotite and pyrite ores contain small quantities of metal sulphides other than iron such as copper, lead, zinc, nickel, cobalt, etc. These metals react in leaching and become constituents of the leach liquor. It was assumed at the outset that these impurities would have a decided effect on the nature of the deposit, and this assumption was proved early in the investigation. Lead is particularly harmful in its effect on the deposition of iron. Its effect varies with its concentration in the electro-

lyte. In concentrations of 0.5 gramme per litre and over, characteristic highly porous, soft, spongy, and dark deposits are obtained. In concentrations below this, down to 0.004 gramme per litre, the deposits are mossy and soft and spongy, with cracking and scaling at the lower limits. Concentrations around 0.002 grammes per litre apparently have no effect on the deposition, provided the rate of flow through the cell does not exceed 4 to 5 litres per hour per square foot cathode area.

Copper in amounts over 0.05 gramme per litre tends to cause scaling and treeing, but in lower concentrations does not materially affect the nature of the deposit although the presence of copper will detract from the value of the iron for many purposes.

Zinc, the most difficult of the impurities to remove from the electrolyte, fortunately is the least disturbing of the elements in its effect on electro-deposition of iron. In concentrations of from 1 to 2 grammes per litre it has the effect of producing pitted and soft deposits, and is readily co-deposited with the iron. Concentrations below 0.35 gramme per litre have no material effect on the nature of the deposit, nor at this concentration does there appear to be any serious co-deposition of zinc and iron. It has not been definitely determined to what extent nickel and cobalt affect deposition, but it may be said that in low concentrations, 0.05 gramme per litre or less, their effect is negligible.

Hydrogen sulphide present in the electrolyte causes a black coated deposit, but does not otherwise seem to affect the nature of the deposit except from a chemical standpoint, when the probability is that the sulphur content will be above the average. Ferric chloride in concentrations of 1 gramme per litre upward causes cracking of the deposit with subsequent scaling or peeling.

PURIFICATION OF ELECTROLYTE

In electro-deposition work, generally, purification of the electrolyte has proved the most difficult problem, and iron deposition is no exception to this rule. Electrolyte liquors obtained from leaching pyrite or pyrrhotite are essentially the same, consisting of ferrous chloride and the chlorides of any other metals present in the ore. The process of purification of electrolyte should, therefore, be very much the same for both ores, the final result sought being a comparatively pure solution of ferrous chloride. The leach liquor obtained from these ores may contain one or more of the following metal chlorides: copper, lead, bismuth, antimony, arsenic, zinc, nickel, cobalt, selenium, calcium, magnesium. The latter two may be disregarded as having no effect in normal quantities, excepting, perhaps, an effect on crystalline structure. The others cause poor deposition or result in iron of less purity, through co-deposition. In the ores used in this investigation, the abundant impurities have been copper, lead, and zinc, with only traces of bismuth, arsenic, and nickel, consequently the three former impurities have received most attention.

Sponge iron at once suggests itself for the removal of copper and lead. In several experiments sponge iron was added to the ferrous liquor in quantities from two to ten times the amount necessary to cement out the copper and lead. This resulted in the removal of 90 to 95 per cent of the copper, but only about half of the lead. Deposition tests on the liquor thus treated gave characteristic lead-iron deposits, and analyses showed the presence of lead and copper. Therefore this method is not to be con-

sidered as satisfactory for removal of small quantities of copper or lead, but might readily be used where the copper content in the liquor is above 3 grammes per litre, 90 per cent of the copper being removed in a marketable form, and the remainder removed by a precipitation process.

Purification

Most of the metals likely to occur in the iron sulphide ores and subsequently in the leach liquors will react with hydrogen sulphide to form insoluble sulphides. Experiments were therefore carried out using hydrogen sulphide, iron sulphide (artificial) and calcium sulphide. Hydrogen sulphide readily precipitates copper and lead in concentrations above 0.02 gramme per litre, quantities smaller than this precipitate with difficulty. Zinc can not be completely removed by hydrogen sulphide and when present in quantities as high as 5 grammes per litre in almost neutral solution (p_H 4) as much as 0.5 gramme per litre may be left unprecipitated. This is probably due to the increased formation of hydrogen ions in solution from the hydrogen sulphide as a result of the sulphide precipitation, resulting in a lesser ionization of the hydrogen sulphide and the establishment of a solubility equilibrium between the zinc sulphide and the solution. The addition of ammonium acetate or sodium acetate to the solution reduced the concentration of the hydrogen ion and the ionization of the hydrogen sulphide takes place to a greater extent, and it is possible by this means to precipitate completely the zinc as sulphide. This method is, however, not considered practical as the amount of acetate required would be quite excessive, resulting in the formation of a complex electrolyte, with the chances of having a high-carbon iron product on electrolysis.

Pure iron sulphide was tried as a precipitant for lead and copper and was found to be fairly satisfactory and might readily be used where these impurities alone occur in an electrolyte. It was added in powdered form, (60 to 80 mesh) to the slightly acidified liquor and resulted in the precipitation of the copper and lead in sufficiently complete amount to make a good electrolyte. Little or no zinc is precipitated by this means. Calcium sulphide as a precipitant has given varying results, and although copper and lead have been fairly consistently and successfully removed from the electrolyte by its use, the same cannot be said of zinc. Liquors containing 0.5 gramme of zinc per litre and upwards have been treated by this method and in all cases zinc has been precipitated, but has never been completely removed. The lowest content yet reached has been 0.15 gramme per litre, but this has only been obtained after two or more treatments with calcium sulphide and with a co-precipitation of iron, resulting in a reduction of the iron content, and the building up of calcium salts in the electrolyte. It would be possible, of course, to recover the precipitated iron and also to eliminate the calcium salts by the addition of ferrous or ferric sulphate in the circuit, but whether this would be an economical procedure has not been determined.

It may be said that the elimination of lead and copper and possibly other heavy base metals, such as bismuth and antimony, presents no great difficulty, but the elimination of zinc still presents a problem.

It would appear, however, from data obtained, that the reduction of the zinc content to 0.3 gramme per litre would be satisfactory, as electrolytes with this zinc content have given iron deposits practically free from zinc.

ELECTRO-DEPOSITION

Electro-deposition tests were carried out in a cell constructed of ebony asbestos wood, 12 inches long, 12 inches deep, and $7\frac{1}{4}$ inches wide, inside dimensions. The cathode, a cylindrical steel mandrel, is separated from the graphite anode plates by an asbestos cloth diaphragm. The cathode has a deposition surface of approximately 1 square foot, and is rotated at a speed varying from 250 to 350 r.p.m.

Electrolysis is carried out with a hot electrolyte and for this purpose a pre-heater was used to feed hot electrolyte to the cell. The pre-heater was made from a piece of Shelby steel tubing 1 inch in diameter, and 2 feet long, surrounded by an insulated wire resistance element; the liquor from the feed tank was run through the heated tube to the cell. This scheme did not work satisfactorily as it was difficult to obtain a continuous flow of definite temperature. Recourse was made to an immersion type of heater controlled by a rheostat and placed in the cell. This consisted of a piece of pyrex tubing in the shape of a broad U having a heating element of nichrome wire spirally wound, running through it and consuming about $\frac{1}{2}$ kilowatt per hour. This heater worked very satisfactorily and stood up remarkably well. The cell temperature was consistently maintained between 80 and 90°C. with the feed rate about 4 litres per hour of cold liquor. The current density usually employed in the deposition tests was 100 amperes per square foot, at a cell voltage of 4.0 to 4.4. In all some forty deposition tests were made using pure and impure electrolytes. The effect of impurities has already been mentioned, and it will only be necessary to outline here a few facts relative to best conditions for satisfactory iron deposition.

The governing factors in cell operation, given a pure electrolyte, may be stated as follows:—

Temperature of electrolyte should be between 80 and 90°C. Lower temperatures incline toward nodular deposits and lower current efficiency. Acidity of electrolyte is also important, high acidity, p_H 3.2 or higher, results in cracking, treeing, and pitting, due probably to high excess of hydrogen at the cathode. The current efficiency is also greatly reduced, the voltage rising with the acidity, and the resultant iron is extremely brittle because of the high hydrogen content. Iron content of the electrolyte does not appear to be a critical factor, good deposits being obtained from concentrations varying from 150 to 220 grammes of iron per litre. An anolyte overflow analysing half ferric and half ferrous would seem the best practice. If the ferric is allowed to build up beyond this, excessive chlorination results which means loss of ferric chloride. A hydrostatic head must also be maintained in the cathode compartment to prevent ferric chloride seeping through into the catholyte. A near neutral electrolyte, p_H 4.0 to 4.8 tends to hydrolyse readily, a considerable precipitate of iron oxychloride being formed. The presence of oxychloride does not adversely affect the nature of the deposition, in fact, it probably acts as a depolarizer, but it impregnates the diaphragm, setting up a resistance and also results in a loss of iron in the solution.

Summing up, the best operating conditions appear to be: temperature 80-90°C., acidity p_H 3.7 to 3.9; iron content 170 to 185 grammes per litre; absence of ferric in catholyte; and pure electrolyte. With these

conditions and proper cell design, current (cathode) efficiencies averaging 94 per cent appear quite possible. The anode efficiency is always somewhat below this, being around 90 per cent, due to slight chlorination and other causes. The following data may be given as an example of a cell run:—

Cathode.....	Steel mandrel 12 x 3½ inches diameter, rolled surface.
Anodes.....	Graphite plates, 12 x 12 x ½ inch.
Electrolyte.....	FeCl ₃ —156 grms. iron per litre.
Feed rate.....	3.4 litres per hour.
Average temperature.....	85° C.
Time.....	5 hours.
Average C.D.....	106 amperes (124 amps. per sq. ft.).
Average volts.....	3.9.
Total ampere hours.....	529.8.
Weight of deposit.....	518 grammes.
Thickness of deposit.....	0.030 inch.
Cathode efficiency.....	93.8 per cent.
Anode liquor.....	61.2 grammes iron per litre.
Anode efficiency.....	90 per cent.
Deposit.....	Silver grey, perfectly smooth, satin surface, hard and brittle, smooth at edges.

The deposit was annealed at about 600°C. and removed from the mandrel as a perfect tube. Analysis was not made on this iron, but other samples from similar electrolyte showed maximum impurities to be lead, copper and sulphur, each less than 0.01 per cent, with carbon 0.002 per cent and traces of phosphorus and zinc, with iron running 99.94 to 99.97 per cent.

The following briefly summarizes the results of 37 deposits run on electrolyte from pyrrhotite and pyrite:—

Excellent.....	Smooth and matt.....	10
Good.....	Slight treeing at edges.....	6
Fair.....	Slightly uneven and coarse.....	8
Poor.....	Treed edges and nodular.....	7
Very poor.....	Cracked and treed.....	6

MANDREL TREATMENT

The condition of the mandrel would seem to be an important point. Best results have been obtained with a machine-turned mandrel subsequently rolled to a smooth surface. Difficulty has been experienced in successfully stripping the deposits from the mandrel, and no definite procedure can be laid down in this regard. Six excellent deposits have been stripped as tubes, and several deposits have been stripped incompletely, or as plates. The problem would seem to be a mechanical one rather than one of deposition conditions. Coatings of various types have been tried, such as oils, grease, metal soaps, etc., but most satisfactory results have been obtained with an extremely thin coating of vaseline.

SUMMARY

The results obtained in this investigation would indicate that the hydrometallurgical treatment of pyrite and pyrrhotite is possible from an electrochemical standpoint. Sufficient data, however, have not been obtained to warrant commercial application, and a continuation of the investigation will be made in an effort to clear up some of the uncertain factors.

Report No. 238

A HYDROMETALLURGICAL TREATMENT FOR PYRRHOTITE OF LOW GOLD
AND COPPER CONTENT, FROM NORANDA MINES, LIMITED,
ROUYN TOWNSHIP, QUEBEC

R. J. Traill and W. R. McClelland

The following tests were carried out on a small shipment of pyrrhotite from the Noranda property with the object of recovering the iron sulphur, gold, and copper present in the ore. The ore consisted of a mixture of pyrrhotite and pyrite with low values in copper, gold, and zinc. The analysis of the ore was as follows:—

Total iron.....	50.3 per cent
Ferrous iron.....	33.18 "
Copper.....	1.20 "
Zinc.....	0.37 "
Gold.....	0.10 oz./ton

Apparently about one-third of the iron content of the ore is present as pyrite which is practically insoluble in ferric chloride solution, and for favourable recovery of iron it would be necessary at some stage in the process to convert this pyrite to artificial pyrrhotite. Two possible methods of treatment were tried, both giving about the same results in iron and copper extractions. The results obtained although not entirely satisfactory are interesting and encouraging from a hydrometallurgical viewpoint and warrant further investigation. Sufficient ore was not available at the time to continue the tests. The following is a brief summary of the methods of treatment and the results obtained.

PRELIMINARY TEST—DIRECT LEACHING METHOD

The ore ground to pass 200 mesh was leached with a hot ferric-ferrous chloride solution. The volume of leaching liquor was 4 litres made up of 70.2 grammes per litre ferric iron and 72.8 grammes per litre ferrous iron. The charge to this leach was calculated on the theoretical requirement based on the soluble (ferrous) iron in the ore, plus a 25 per cent excess, and amounted to 528 grammes ore. When the leach liquor was completely reduced the residue was separated by filtration, dried, and weighed. The residue weighed 378 grammes and contained 14.56 per cent soluble iron. This residue was submitted to a non-oxidizing roast using an iron pipe retort at 750°C, the elemental sulphur resulting from the leaching and one atom of sulphur from the pyrite being distilled off. The roasted product weighed 241 grammes and analysed 54.6 per cent soluble iron, showing that the pyrite had been changed from an insoluble to a soluble artificial pyrrhotite. This product was then submitted to a further leaching with ferric-ferrous chloride and a residue obtained which weighed 134 grammes and analysed 17 per cent total iron. Distillation of the elemental sulphur from this residue resulted in a product weighing 73 grammes, of the following analysis:—

Total iron.....	31.40 per cent
Ferrous iron.....	16.40 "
Copper.....	1.16 "
Gold.....	0.78 oz./ton

It will be noted here that the gold value has been concentrated approximately 1 : 8. The presence of 1.16 per cent copper in this final residue may preclude the practicability of direct treatment of the residue for gold recovery by cyanidation. The high iron and copper content is probably due to the presence of unaltered chalcopyrite and pyrite and a further non-oxidizing roast followed by leaching would probably result in a residue that could be economically treated by cyanidation for gold, or the residues could be sent to a smelter for treatment for recovery of gold and copper. The total extractions from the ore obtained by this method show:—

Iron extraction.....	91.3 per cent
Copper extraction.....	86.6 “

DIRECT ROASTING METHOD

In an effort to shorten the process direct roasting of the ore was attempted, with the expectation that the pyrite would be altered to artificial pyrrhotite and that the natural pyrrhotite would remain unaltered. This would seem a feasible theory, but results proved unsatisfactory in that the iron extractions were very low, due it is thought to an alteration of some nature in the pyrrhotite by roasting causing excessive hydrolysis in the leaching. This oxychloride of iron formed by the hydrolysis slowed up the reaction between the ore and the ferric chloride quite seriously, and it was necessary to add a greater excess of ore to bring about complete reduction of the liquor. This resulted in lower percentage extraction and high iron content residues necessitating further roasting and leaching to obtain a favourable recovery of iron or copper. The final result was somewhat similar to that obtained by direct leaching, the residue having a high iron and copper content, and the gold concentrated about 1 : 8. This test was carried out on a larger scale, using the roasting furnace leaching and cementation equipment as described in the report on pyrrhotite treatments, and finally depositing the iron in a diaphragm cell. The following is a brief summary of the results obtained: 13 kilograms (approx. 30 pounds) of the ore was ground to 10 mesh, fed to the rotary furnace and roasted under non-oxidizing conditions at 750°C. One pass through the furnace was made resulting in a product weighing 12.24 kilograms, and analysing:—

Total iron.....	52.42 per cent
Ferrous iron.....	48.96 “
Copper.....	1.28 “

A small amount of sulphur was recovered, but most of the sulphur set free was oxidized in the furnace to SO₂. The roasted product was leached in 3 lots with hot ferric-ferrous chloride solution, each leach having a volume of 50 litres. The leaching action, as already stated, was slow due to excessive hydrolysis. The residue from leaching weighed 7.9 kilograms and the extraction of iron and copper amounted to:—

Iron extraction.....	59.7 per cent
Copper extraction.....	64.4 “

The leach residue was subjected to distillation, resulting in the recovery of 1.8 kilograms of sulphur. The residue from distillation weighing 5.3 kilograms and containing approximately 50 per cent total iron, was given a secondary roast to convert any unaltered pyrite to pyrrhotite, and a product weighing 4.78 kilograms was obtained.

Total iron.....	54.6 per cent
Ferrous iron.....	53.8 “
Copper.....	1.13 “

One hundred and fifty-two grammes of sulphur was recovered. The product from the second roast was leached with a ferric-ferrous chloride solution and a residue weighing 2.74 kilograms obtained, which on being subjected to a sulphur distillation roast yielded a final residue of 1.6 kilograms.

Total iron.....	43.2 per cent
Ferrous iron.....	38.1 "
Copper.....	1.14 "
Sulphur.....	28.5 "
Gold.....	0.76 oz./ton

The residue obtained, 1.6 kilograms, would show a concentration by weight of 8.15 : 1. The gold content in the head sample amounted to 0.0015 ounce, and in the final product 0.0013 ounce, showing a slight loss which may have occurred in handling. A summary of these results showed:—

Total iron extraction.....	89.5 per cent
Total copper extraction.....	88.6 "
Sulphur recovered, 2.7 kilograms.....	60.4 "

PURIFICATION OF ELECTROLYTE

The leach liquors from the primary leaching of the direct roasted product analysed about 163 grammes per litre ferrous iron, about 0.6 gramme per litre copper and 0.4 gramme per litre zinc, and the leach liquor from the secondary leaching analysed 145 grammes per litre ferrous iron, 0.7 gramme per litre copper and less than 0.2 gramme per litre zinc. In the purification of this liquor an attempt was made to separate the copper by the addition of sponge iron. This, however, was only partly successful, the last traces of copper being difficult to remove, and recourse was had to the use of calcium sulphide to remove the remaining copper. This procedure resulted in an electrolyte of sufficient purity for electro-deposition of iron, the only apparent impurity being zinc, which was present to the extent of about 0.35 gramme per litre.

ELECTRO-DEPOSITION OF IRON

Electro-deposition was carried out in the diaphragm type cell, the cathode being a steel mandrel of approximately 1 square foot area, the anodes being graphite plates. An asbestos cloth diaphragm separated the anolyte from the catholyte. A description of the cell and its operation is outlined in the report on pyrrhotite and pyrite. The following summary of cell operation data serves as a typical example of the tests on this electrolyte:—

Electrolyte—FeCl ₂	145 grammes per litre Fe.
Rate of electrolyte feed.....	3.8 litres per hour.
Time.....	5 hours.
Average current density.....	103.7 amperes per sq. ft.
Average voltage.....	3.7.
Average temperature.....	86° C.
Weight of deposit.....	490 grammes.
Thickness of deposit.....	1/32 inch (approx.).
Speed of cathode rotation.....	274 r.p.m.
Anode overflow, Ferrous.....	76 grammes per litre.
Ferric.....	60 grammes per litre.
Current efficiency.....	90.8 per cent.
Nature of deposit.....	Silver-grey in colour, surface smooth with a very few small nodules, very finely crystalline, coherent and of uniform thickness. After annealing at 400 to 600 ° C. to remove hydrogen, the deposit was removed from mandrel by rolls as a tube, perfect in shape.

RECOVERY OF GOLD IN RESIDUE

The residues obtained from leaching were submitted to cyanidation. J. S. Godard carried out the tests and reported results as follows:—

Residue grms.	Solution c.c.	KCN per cent	Assay head	Tailing oz./ton	Extraction per cent	KCN con- sumed, lb./ton	Time hours
606	2360	0.2	0.76	0.185	75.6	32.7	50

Grinding.—Residues ground wet in a ball mill using grey iron balls. Intermittent screening through 200 mesh. Oversize returned to mill.

Remarks.—Cyanidation in a more dense pulp 1:2.5 using 0.075 per cent cyanide and increasing the time might increase the extraction with a considerable decrease in cyanide consumption. The residues contained 1.14 per cent copper and with a more complete removal of the copper the gold extraction should be better, and the consumption of cyanide much lower. With a smelter in close proximity, cyanidation of the residues would not be necessary, as this product contains sufficient values in gold and copper to be shipped to a smelter for recovery of the values.

SUMMARY

Two methods of treatment are described namely (a) directly leaching the pyrrhotite content, and with subsequent roasting of the pyrite; and (b) directly roasting the mixed sulphides thereby converting the pyrite to artificial pyrrhotite. The latter method proved less satisfactory than anticipated, a new problem arising in the roasting effect on pyrrhotite, necessitating further roasting treatment to secure satisfactory iron recovery. Both methods give practically the same results.

Results show extractions of iron and copper amounting to 89.5 per cent and 88.6 per cent respectively, with 60.4 per cent recovery of sulphur under more or less adverse conditions.

Concentration of gold values in the residue in the ratio of 1:8 results from either method of treatment.

A pure electrolyte can be obtained, suitable for recovery of iron as electrolytic iron.

Report No. 239

A NEW PROCESS FOR THE TREATMENT OF ILMENITE FOR THE RECOVERY
OF ELECTROLYTIC IRON AND TITANIUM OXIDE CONCENTRATE
FOR PIGMENT AND OTHER PURPOSES

R. J. Traill and W. R. McClelland

Microphotographs by E. A. Thompson

INTRODUCTORY

Canada has, in the province of Quebec, deposits of ilmenite which compare favourably as to size and grade with any of the known deposits in other countries. These deposits are favourably situated with respect to transportation, to the use of cheap hydro-electric power for their reduction, and to general markets for the disposal of commercial products obtained from them. No industry has been established in Quebec for the utilization of these ores. Production has been confined to shipments of ore to United

States for the manufacture of ferro-titanium and titanium pigment. The latter is a recent development, the result of Norwegian chemical research, and it promises to be a most valuable one, and should develop into an important commercial industry in Canada. The special qualities of titanium white pigment may be briefly summarized as follows:—

1. Greater covering power than zinc oxide or white lead.
2. Chemically inactive with vehicles or other pigments.
3. Not affected by acid fumes or sulphide gases.
4. Remains white under any atmospheric conditions.
5. Is non-toxic.

Possessing such good qualities, it is highly probable that this pigment will replace white lead or zinc oxide to a great extent.

In the present practice of pigment manufacture from ilmenite ores the titanium content alone is recovered, the iron content averaging 30 to 40 per cent going to waste. A process that would recover both the iron and titanium content would therefore be advantageous as it would result in a fuller utilization of the valuable constituents of the ilmenite ores of the country.

A process having this end in view has been investigated in the electro-chemical laboratory and the results obtained have been very encouraging and warrant further investigation on a semi-commercial or pilot-plant scale to check and corroborate the results, and to determine cost data. A brief summary of the process follows together with observations respecting important details of operation which should be of material assistance in following these tests up on a somewhat larger scale. The products of this process are electrolytic iron and titanium oxide concentrate suitable for pigment-making purposes. There is a considerable market for electrolytic iron, and its physical properties are now sufficiently well known to assure a steady and increasing demand.

THE PROCESS

The ilmenite is reduced by means of carbon in the form of coal, coke, or charcoal, in a furnace in which the temperature and atmosphere can be controlled. The iron oxide is converted into spongy metallic iron, but the titanium oxide is chemically unchanged. The product is passed over a magnetic separator to remove unburnt carbon and dissociated gangue material. The magnetic product, consisting almost entirely of metallic iron in the form of sponge, and titanium oxide is then treated with ferric chloride solution. The iron is dissolved, forming ferrous chloride, leaving an insoluble residue of impure titanium oxide which is separated by filtration or other means, and subsequently treated by an acid or acid sulphate process for recovery of pure oxide of titanium. The ferrous chloride liquor obtained after separation of the insoluble residue may contain impurities, and it is purified at this stage by a precipitation process which eliminates the undesirable elements. Electro-deposition of iron is carried out in a diaphragm cell, the iron being deposited on a rotating, steel-mandrel cathode, or steel plates, and with anodes made up of graphite. With the deposition of iron in the catholyte compartment, oxidation in the anolyte compartment takes place forming ferric chloride, which is returned to the leaching system for action on fresh sponge. The mandrel with the deposit of iron is then given a pre-annealing treatment to drive



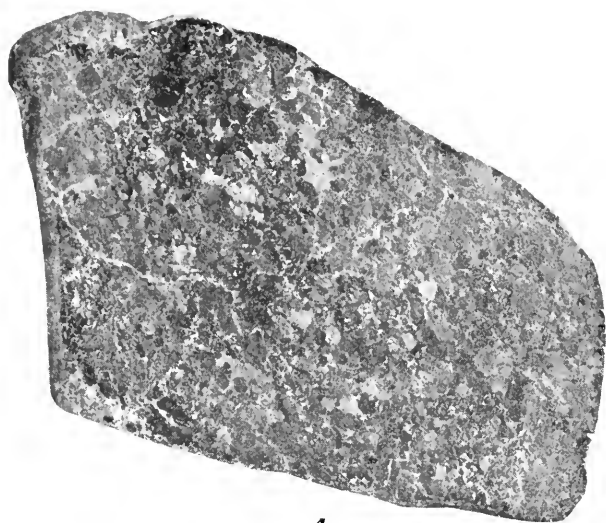
Plate III

- A. Polished specimen of lavy ilmenite ore. The ore is dense black, medium even-grained, granular ilmenite with some hematite, carrying scattered particles of iron pyrite, occasional flakes of black mica, a few grains of smoky quartz, and a little feldspar. It is feebly magnetic and has a reddish black streak. No magnification.
- B. Polished specimen, unetched. Light streaks are hematite in the darker ilmenite. Black spots are gangue. The lamellae of the ore have been displaced by minute faults. Magnification, 200 diameters.
- C. Polished specimen, etched with strong hydrochloric acid. Shows three grains in contact differentially orientated. The dark streaks are hematite, etched out, and the light ilmenite. The dark spots are gangue. Magnification, 200 diameters.

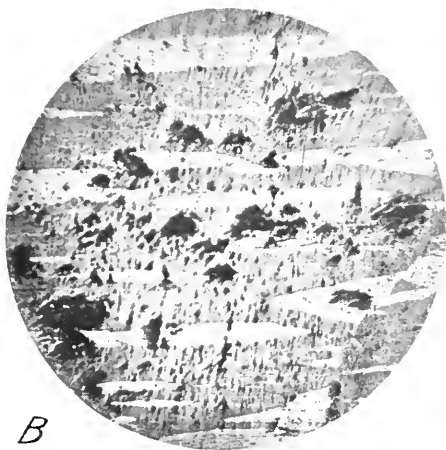


Plate III

- A. Polished specimen of Ivry ilmenite ore. The ore is dense black, medium even-grained, granular ilmenite with some hematite, carrying scattered particles of iron pyrite, occasional flakes of black mica, a few grains of smoky quartz, and a little feldspar. It is feebly magnetic, and has a reddish black streak. No magnification.
- B. Polished specimen, unetched. Light streaks are hematite in the darker ilmenite. Black spots are gangue. The lamellae of the ore have been displaced by minute faults. Magnification, 260 diameters.
- C. Polished specimen, etched with strong hydrochloric acid. Shows three grains in contact differentially orientated. The dark streaks are hematite, etched out, and the light, ilmenite. The dark spots are gangue. Magnification, 260 diameters.



A



B



C

out the hydrogen, and the deposit is removed from the mandrel, by suitable rolling device, in the form of a tube. The tube may be further annealed to suit cold rolling work.

EXPERIMENTAL WORK

Sponge Making

Ore.—The ore used was ilmenite from Ivry, Que., on the Mont Laurier branch of the Canadian Pacific railway. It was submitted by Messrs. McArthur, Irwin, Limited, of Montreal, and contained as shown by analysis:—

Insoluble.....	7.90 per cent
TiO ₂	30.80 "
Fe.....	40.70 "

Furnace.—In the making of sponge iron the ideal practice would be one embracing continuous operation, and a rotary furnace system would probably best meet this requirement. Although a small oil-fired rotary furnace was available, tests conducted some years ago on metallization of hematite ore in this furnace gave results which were unsatisfactory on account of the difficulty of maintaining a reducing atmosphere, and therefore, recourse was had to a brass-melting, pot furnace, gas-fired, for the making of the sponge iron. The United States Bureau of Mines, at their Seattle station, have developed a rotary kiln furnace, oil-fired, which seems to have given good results on the metallization of iron oxide ores, hematites and magnetites. The furnace with which they conducted their experiments had a capacity of 1,400 pounds of sponge iron per day, and proved to be quite satisfactory for continuous operation. Messrs. Thornhill and Anderson have developed a furnace of the revolving hearth, muffle type for the metallization of iron oxide ores.

Mix, Ore, Temperature, and Time.—The charge to the fireclay pots consisted of three parts ilmenite, 60 to 80 mesh, to one part charcoal, 10 to 20 mesh, by weight, thoroughly mixed. The pots, holding 32 pounds mix, with a slight covering of charcoal were placed in the furnace at a temperature of about 900 to 1,000° C. The heat penetration through such a mixture is necessarily slow, requiring five to six hours to reach centre. When the charge reached about 950° C. the temperature was maintained for about 1½ hours. The pots were allowed to cool in the furnace overnight and the product passed through a 10-mesh screen preparatory to magnetic separation which was made on a Wetherill magnetic separator.

Analysis Sponge Products.—An average test shows analysis of products as follows:—

	Total iron	Metallic iron	TiO ₂	C
Magnetic.....	46.7	40.5	36.9	2.0
Non-magnetic.....	12.8	2.7	9.3	37.9
Metallization.....				83.2 per cent
Yield metallic iron.....				80.3 "

With properly designed furnace the yield and metallization of iron can probably be improved. The most important fact determined in these tests is the necessity for careful control of temperature. At temperatures above 1,000° C. sintering of the mass, or fusion, results, and tests would

indicate that such a product is not readily soluble in ferric chloride liquor. Using a rotary furnace, oil-fired, it is probable that the proportion of coal to ore will need to be higher, say 75 : 100, to ensure a completely reducing atmosphere. Much of the excess should, however, be recovered in the magnetic separation, and used in subsequent mixes.

Sponge Leaching

Scale of Tests.—Initial leaching tests were carried out in enamel pails of 6 litres capacity, fitted with agitating devices, the volume of leaching liquor used being 4 litres. Larger leaching tests (50 litres) were carried out in 15-gallon stoneware tanks, similar in design to nitrating kettles, and fitted with stoneware agitators. The leaching liquor used was a mixture of ferrous and ferric chlorides, made up from ferric chloride salt partly reduced with iron, or from liquor obtained as anolyte overflow from the electro-deposition cell. The charge of sponge for leaching was dependent upon the amount of ferric iron in the leaching liquor, sufficient being added to cause complete reduction of all iron to the ferrous state. The charge is calculated from the equation, $2\text{Fe Cl}_3 + \text{Fe} = 3\text{Fe Cl}_2$.

Temperature of Leaching.—Leaching was carried out with both cold and hot leaching liquor to determine the difference, if any, in time and extraction factors. Leaching at a temperature between 90 and 100° C. results in a quicker reaction between the sponge and liquor, with a higher extraction of metallic iron, recoveries running 98 to 99 per cent. The time factor is reduced by more than one-half. The titanium content is, however, somewhat higher than obtains in cold leaching (0.0375 gramme per litre as against 0.015 gramme per litre in cold leaching). Cold leaching should average about 92 to 95 per cent extraction. It may be stated here that the reaction $\text{Fe} + 2\text{Fe Cl}_3$ is exothermic and that a temperature of about 50° C. would be reached starting with initial temperature of 20° C. It would be difficult to say which treatment causes the greater hydrolysis of ferric chloride, and whether anything is gained by hot leaching.

Filtration.—Suction filtering was used in separating insoluble residue from leach liquor, but proved somewhat slow, especially as filter cake thickened. A combination of settling or thickening and filtering would probably be more satisfactory.

Impurities.—The Ivry ore contained small amounts of copper, lead, nickel, and cobalt, and these elements were found in the leach liquor. As they exert a harmful effect on the electro-deposition of iron they were precipitated as the sulphides by the use of calcium sulphide CaS , in the presence of a small amount of acid removed by filtering and subsequently recovered by known processes.

Selective Leaching and Purification.—It is possible, however, to partly overcome this step by using an excess of sponge material in leaching which would prevent the copper and lead, which are the most harmful from the deposition standpoint, from going into solution. The liquor or electrolyte would then contain only small amounts of nickel and cobalt which would co-deposit with the iron without seriously affecting its general properties. Such a method would of course necessitate further treatment of the residue by excess ferric chloride solution to completely remove the excess of iron and free the copper and lead from the titanium oxide. This

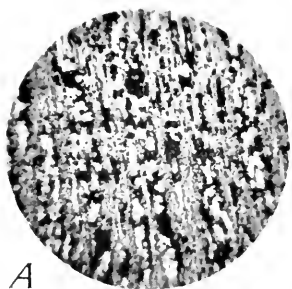


Plate IV

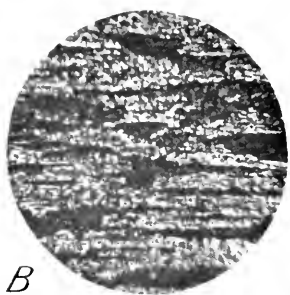
- A. Mounted specimen of metalized ore, unetched. The honeycombing and fine, dark specks are due to the removal of oxygen, leaving the titanium oxide and sponge iron in white and grey relief. The large, dark specks are gangue. Magnification, 200 diameters.
- B. Mounted specimen of titanium oxide residue, unetched. The clear white skeleton is titanium oxide after the removal of the greater proportion of the iron by leaching. The dark bands are gangue. Magnification, 200 diameters.
- C. Polished specimen of electrolytic iron, not annealed or rolled. Etched with 10 per cent nitric acid and 90 per cent alcohol for 10 seconds. Magnification, 325 diameters.
- D. Polished specimen of electrolytic iron, annealed at 600 to 600° C. Etched with 10 per cent nitric acid and 90 per cent alcohol for 10 seconds. Magnification, 325 diameters.
- E. Polished specimen of electrolytic iron, rolled, and annealed at 1,000° C. Etched with 10 per cent nitric acid and 90 per cent alcohol for 10 seconds. Magnification, 425 diameters.

Plate IV

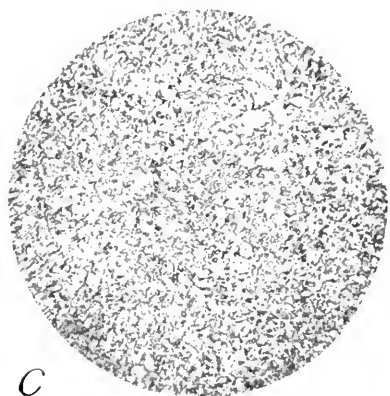
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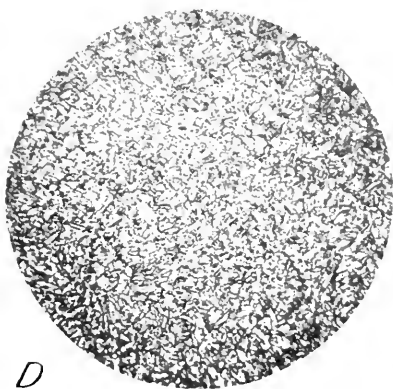
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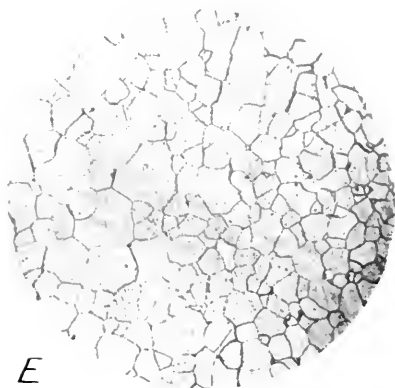
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C



D



E

leach liquor could then be treated for recovery of copper and lead which could be built up through successive residue treatments and recovered more readily in this higher concentration, by cementation or other method.

Electro-deposition

Cell.—The type of cell used in these experiments was a diaphragm cell with an electrolyte capacity of about 12 litres, having a steel mandrel of one square foot plating area as cathode, and two graphite anode plates. The catholyte and anolyte cells are of almost equal capacity and separated by an asbestos diaphragm of U-shape, the cathode being inside the diaphragm and the anodes being placed down the sides and across the bottom close to the diaphragm. Outlets on the anolyte sides of the cell were provided for drawing off the regenerated ferric chloride, and an outlet from the catholyte was arranged to regulate the hydrostatic head necessary for proper operating conditions.

Electrolyte Feed.—The electrolyte was fed into the catholyte compartment at a regulated rate which averaged 4 litres per hour. Satisfactory deposition can only be made at temperatures between 80 and 95° C. This necessitates pre-heating the electrolyte before going to the cell, or arranging a heating system in the cell itself to maintain the temperature required. The speed of the rotating mandrel should be somewhere between 250 and 400 r.p.m.

Preparation of Mandrel.—The preparation of the mandrel is highly important and although various schemes have been tried out, no single method has proved satisfactory with regard to successful stripping of the deposit. The removal of the deposit is of course strictly a mechanical problem and with proper conditions and apparatus should be readily accomplished. The obtaining of good smooth deposits, however, depends upon a smooth surfaced mandrel. Coatings of various types have been tried but vaseline seems to be the most satisfactory. Smoothed mandrels having iron rust coatings have also given good results.

Current Density and Voltage.—Current densities around 100 amperes per square foot have proved quite satisfactory. The voltage depends on the resistance set up by the electrolyte, the diaphragm, and the distance between electrodes. In the design of cell used in this laboratory an average of 4 volts has been obtained at the above current density.

Effect of Impurities Nickel, Cobalt, Copper, Lead.—The presence of nickel, copper, or cobalt in the electrolyte in quantities less than 0.02 gramme per litre has no apparent effect on the deposit, larger quantities tend to cause cracking or stripping of the deposit. The presence of lead in quantities below 0.002 gramme per litre has no ill effect, but quantities above this cause spongy deposition, pitting and treeing, according to the amount present.

Effect of Low Temperature.—Deposition at 70°C. or lower is attended with nodular formations and generally uneven deposits. The voltage required is also higher, resulting in reduced current efficiency.

Acidity.—The electrolyte should be near neutral pH 3.7 to 3.9 giving best results.

Nature of Deposits.—Deposits should be silver-grey in colour, smooth, and microscopically non-crystalline. Best conditions are pure electrolyte, temperature 85 to 95° C. and careful regulation of C.D., and also acidity not greater than pH 3.7.

Occluded Hydrogen.—The iron as deposited contains hydrogen, thereby making it extremely brittle. This hydrogen is readily removed by slightly annealing at 300° C. upwards.

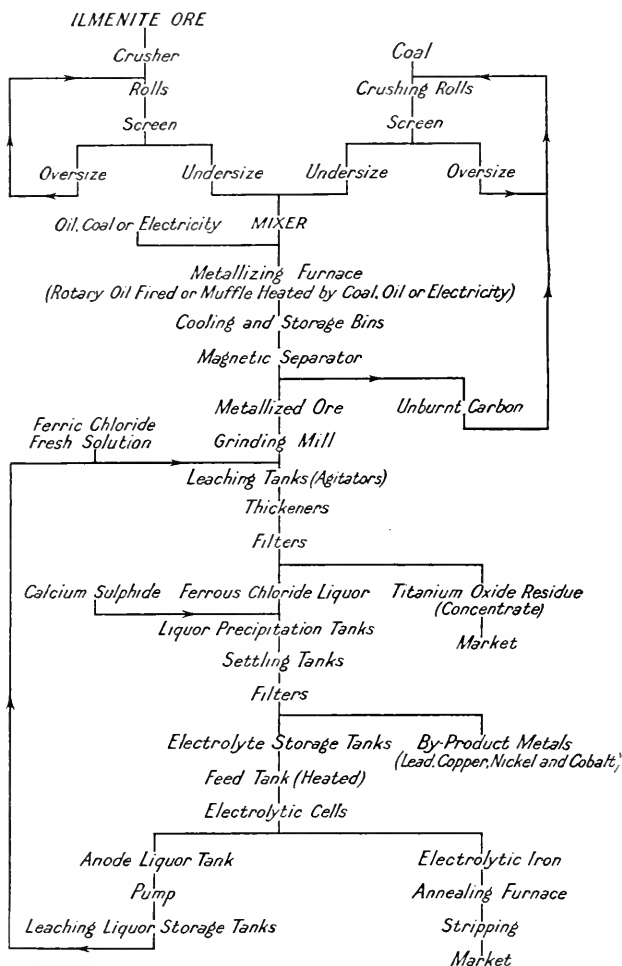


Figure 3. Flow-sheet of process for the production of electrolytic iron and titanium oxide concentrates from Canadian ilmenite ores.

Test Example

Sponge.—The following is submitted as an example of a complete test.

Leaching.—Charge to pot for sponge making consisted of:

Ivry ilmenite.....	3 parts.
Charcoal.....	1 part.
Metallization temperature.....	990° C.
Time.....	1½ hours.
Product 20-lbs—Magnetic.....	16½ lb.
Non-magnetic.....	2½ “
Loss.....	1½ “ (sinter dust, etc.)
Metallic iron in magnetic product.....	35.7 per cent.
Yield metallic iron.....	79.5 “
Metallization.....	82.0 “
Liquor—FeCl ₂	60.5 grammes Fe per litre.
FeCl ₃	71.5 “
Volume.....	50.0 litres.
Sponge iron required to reduce FeCl ₃	71.5 x 50 ÷ 2 = 1,787.5 grammes.
Sponge material added.....	5,000 grammes = 1,785 grammes iron.
Feed rate.....	Complete in 10 minutes.
Time of leaching.....	1½ hours.
Temperature.....	21° C. start, rising to 47°—exothermic reaction.
Iron dissolved.....	1,713.8 grammes (calculated from reactions).
Extraction of iron.....	96 per cent.
Liquor strength.....	167 grammes per litre.
Weight of residue.....	3,415.5 grammes.
Analysis of residue—	
Total iron.....	16.13 per cent.
Metallic iron.....	0.25 “
Titanium oxide.....	53.62 “

NOTE.—Weight of residue 3,415.5 grammes plus calculated weight of iron dissolved, 1,713.8 grammes, totals 5,129.3 against 5,000 grammes feed. This increase in weight may be accounted for as due to hydrolysis of solution, and also to oxidation of undissolved iron in residue during drying.

Assuming this theory to be correct, calculation shows that hydrolysis occurred to the extent of 1 per cent iron content of leaching liquor.

Purification of Electrolyte.—The electrolyte obtained contained copper, lead, nickel, and cobalt, 100 grammes calcium sulphide was added to the cold liquor and after 10 minutes' agitation the precipitate was allowed to settle, and the solution was filtered. The filtered liquor showed a very slight trace of unprecipitated sulphides. The excess of hydrogen sulphide in the liquor was destroyed by aeration. The black precipitate contained: lead, 8.9 per cent; copper, 2.2 per cent; nickel, 1.9 per cent; cobalt, 2.2 per cent; the remainder being iron and sulphur. The liquor strength was 164.2 grammes per litre.

Iron Deposition.—

Cathode.....	Rotary mandrel, steel, approximately 1 square foot area —speed 350 r.p.m.
Anodes.....	Graphite plates 12 x 12 x ½ inch.
Liquor.....	164.2 grammes ferrous per litre.
Time.....	6 hours.
Current density.....	Max. 125, min. 100, amperes per sq. ft.
Average density.....	About 120 amperes per sq. ft.
Total ampere hours.....	714.1.
Voltage.....	Max. 4.4, min. 4.2 volts.
Average.....	4.3 volts.
Temperature.....	Max. 90° C., min. 80° C. Average, 85° C.
Liquor feed.....	22½ litres, or 3.75 litres per hour.
Cathode liquor overflow.....	4½ litres.
Anode liquor overflow.....	17½ litres, approximately 75 grammes per litre ferrous iron.
Weight of deposit.....	702 grammes.
Thickness.....	0.043 inch.
Current efficiency.....	94.3 per cent.
Anode efficiency.....	92.4 per cent.
Nature of deposit.....	Silver-white, finely crystalline, smooth, with a few slight knobs, coherent.

Stripping.—

The deposit was annealed at a low temperature preparatory to stripping. Stripping was attempted in lathe using a roller, but, as deposit loosened only in places, it had to be split and removed as sheet.

Analysis of Deposit.—

		Per cent	Quantity used for determination
Carbon.....	less than	0.001	2 grammes
Manganese.....	"	0.005	5 "
Phosphorus.....	"	0.006	10 "
Sulphur.....	"	0.007	10 "
Silicon.....	"	0.006	10 "
Copper.....	"	0.0024	100 "
Cobalt.....		0.015	100 "
Nickel.....		nil	100 "
Chromium.....		nil	25 "
Titanium.....		nil	25 "
Total impurities.....		0.0424	
Iron, approximate.....		99.96	

Cold Rolling Test.—A piece of the sheet, 0.5 inch wide, 1.0 inch long and 0.043 inch thick, was cold rolled without further annealing to a length of 13 inches and width of 0.64 inch, thickness 0.003 inch, showing:—

Increase in length.....	1,200 per cent
Reduced area.....	91 "
Reduced thickness.....	93 "

The rolled strip was quite sound and showed no cracking or holes. Another sample rolled to 0.002 inch thickness was also found to be sound and free from cracks and holes.

Summary of Test.—

1. A sponge was made from ilmenite ore and charcoal with a metallic iron yield of 79.5 per cent.
2. Leaching this sponge results in a 96 per cent extraction of the iron and the formation of a concentrate of titanium oxide (TiO_2) assaying 53.62 per cent.
3. Small amounts of other valuable metals, namely copper, nickel, cobalt, and lead, were separated in a concentrated form for recovery treatment.
4. Electro-deposition of iron carried out with a current efficiency of 94.3 per cent and a good deposit of iron 99.96 per cent pure.

IV

REPORT OF INVESTIGATIONS: PYROMETALLURGICAL
LABORATORYA COMBINED PYRO AND HYDROMETALLURGICAL PROCESS FOR THE
TREATMENT OF NICKELIFEROUS PYRRHOTITE AND OTHER SUL-
PHIDE ORES FOR THE RECOVERY OF IRON AND SULPHUR
IN ADDITION TO THE OTHER CONTAINED METALS

H. C. Mabee and A. E. Smaill

The small-scale laboratory experiments which were begun in 1924 and described in the Report of Investigations for 1924, page 104, were continued until it was clearly indicated that it would be advisable to enlarge the scale of experiments. The small-scale experiments showed that the most promising procedure after the production of the self-disintegrating matte would be roasting, chloridizing, and leaching. The scale of the experiments was, therefore, enlarged along these lines. This necessitated the construction of a specially-designed chloridizing furnace and the installation of wooden, leaching and filter tanks.

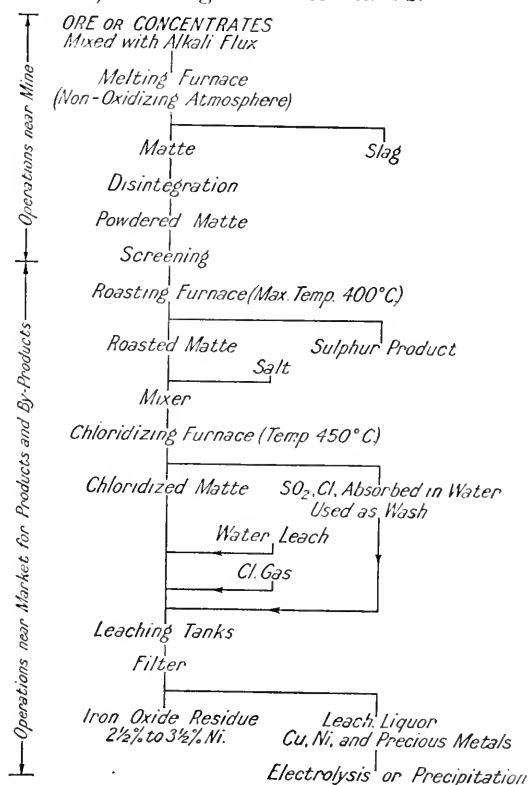


Figure 4. Flow-sheet of process for the recovery of iron and other contained metals from nickeliferous pyrrhotite and heavy sulphide ores.

The flow-sheet, Figure 4, indicates the procedure followed in these tests. A low-grade matte containing practically all the sulphur and metal

values of the ore is first produced. The matte after disintegration is roasted, chloridized and then leached, leaving a residue of the matte in the form of a finely divided oxide. It will be seen that the process is simple and direct, and that marketable products should be obtained in very short actual working time. The losses are very low and the recoveries can be made without the destructive effects on agriculture that occur under present methods. The fuel requirements should be low and where electric power is available it can be employed for the entire process, including smelting.

Matte.—When a sulphide ore is melted with an alkali flux under non-oxidizing conditions, a low-grade self-disintegrating matte containing the sulphur and metal values is produced, the slag carrying the non-metallic gangue and other impurities. The matte for these experiments was made by melting the ore in 10-kilogram lots, in a gas-fired crucible. When cool the slag separated cleanly from the matte by fracture. The concentration in the matte produced is as follows:—

Ore head	Per cent of ore in matte	Copper		Nickel		Iron	
		Head	Matte	Head	Matte	Head	Matte
1. Shebandowan.....	31.5	1.65	5.05	2.97	9.00	24.30
2. " (combined).....	65.5	4.02	7.55	3.82	6.92	31.7
3. "	67.8	4.78	6.89	4.3	5.57	31.7	42.6
4. Frood.....	34.4	1.45	4.00	1.35	3.8	18.79	56.56
5. Murray (tabled).....	68.0	0.92	1.35	1.92	2.95	31.18	51.4
6. "	62.8	0.77	1.02	2.16	2.74	36.42	43.83

The ratios show that this concentration is practically one of elimination of the gangue only:—

Ore	Copper to nickel		Copper and nickel to iron	
	Ore	Matte	Ore	Matte
1. Shebandowan.....	1 : 1.8	1 : 1.78		
2. "	1 : 0.95	1 : 0.92		
3. "	1 : 0.9	1 : 0.81	1 : 3.49	1 : 3.42
4. Frood.....	1 : 0.93	1 : 0.95	1 : 6.7	1 : 7.0
5. Murray.....	1 : 2.08	1 : 2.18	1 : 11.2	1 : 11.9
6. "	1 : 2.8	1 : 2.7	1 : 12.42	1 : 11.6

On exposure the matte readily disintegrates into a finely divided state; about 50 per cent is less than 200 mesh and only about 2 per cent will not pass an 80-mesh screen. There is no segregation of values. The moisture content is less than 3 per cent. An attempt was made to concentrate the nickel and copper values by tabling and magnetic separations but the results obtained were found to be of little practical value. Flotation tests on fresh matte showed a certain amount of concentration but as this method would hardly prove practical the tests were not followed up.

Roasting.—The fine subdivision and the low ignition point of the matte suggest the possibility of reducing the sulphur by combustion in the same manner in which powdered coal is now used. This has not been followed up on a large scale although small-scale experiments were promising and would probably give very uniform results without the use of mechanical stirrers, etc.

The roasting experiments were carried on in a large 4-tier muffle furnace, the muffles being so arranged that the material can be raked from the top muffle down into the next below and so on. The matte was charged in 10-kilogram lots and was stirred at regular intervals. The temperature was kept at about 400°C. until the bottom muffle was reached, where the temperature was not allowed to exceed 600°C. The total time of any batch in the furnace was approximately 4 hours.

Chloridizing Roast.—The roasted material was next given a chloridizing roast to render the copper and nickel more soluble, and further reduce the sulphur. The best experimental results were obtained by roasting in thick beds with moistened matte. To obtain these conditions on a large scale a special chloridizing furnace was designed and constructed. It consists of a vertical cylinder of conical section made of burnt fireclay mixture. The principal internal dimensions of the cylinder are, top 4 inches diameter, bottom 7 inches diameter, height 3 feet. The cylinder is externally electrically heated by means of heavy resistance wires wound vertically, the current being controlled by a suitably tapped transformer. The heating elements are surrounded by a thick heat insulation made of a porous fireclay mixture, burnt in place, with a kieselguhr backing and held in place by a covering of sheet iron. The cylinder is open at the bottom but the top is closed by a cover with openings through which thermo-couples can be inserted. Outlet holes in the cylinder connected to suction are provided just below the cover, by means of which the evolved gases are drawn through a water-cooled section, or condenser, fitted with a trap at the bottom, then through two 20-litre wash bottles before going to waste. The liquor caught in the trap is largely hydrochloric acid carrying some iron, nickel, and copper, some of which if not all is due to matte particles being drawn in and dissolved. The absorption of the sulphur gases by the wash bottles is improved by the cooling in the condenser section of the suction system.

Charge.—The total amount in the furnace when fully charged was approximately 15 kilograms. The chloridizing operation was continuous, but at the same time intermittent in that following a pilot charge of 5 kilograms, held up in place with a charge of quartz, space was made for a further addition of a 5-kilogram batch by raking out sufficient at the bottom, the whole column dropping down. Each batch consisted of 5 kilograms of roasted matte well mixed with about 10 per cent of sodium chloride and dampened with sufficient water to cause the material to retain its form when moulded by hand. A thermo-couple was inserted well into the fresh charge and as soon as the temperature rose to about 400°C., raking out at the bottom was started. The time for a batch to heat to this temperature varied with the condition of the furnace, but when in good running condition, was approximately one hour. The water in the wash bottles when saturated with sulphur vapours was found to be a very effective leach for the copper.

Leaching.—The leaching was accomplished in an elevated cypress wood tank, 3 feet in diameter, equipped with mechanical stirrers, and a means for injecting a small quantity of steam when heating was desired. From the leach tank the batch was washed on to a filter bed equipped with air suction. The leach liquors finally reached a portable cypress wood storage tank which could also be used for electrolyzing.

The total amount of chloridized matte leached in the first trial was 77.7 kilograms. This was leached with absorption water from the wash bottles of a previous chloridizing roast followed by a 10 per cent sulphuric acid wash. The resulting dried residues weighed 56.2 kilograms, the analysis of which showed:—

Iron.....	64.4	per cent
Nickel.....	3.3	"
Copper.....	0.36	"
Sulphur.....	0.21	"

The nickel to iron ratio is 1:19.3. This would make a nickel-steel of 5.17 per cent. The nickel-copper to iron ratio is 1:17.45, which would make a steel carrying 5.72 per cent copper-nickel.

The residues are finely divided oxides and a screen analysis shows:—

Mesh	Per cent
—200	48.6\
—100+200.....	30.3\
— 65+100.....	11.1\
+ 65.....	10.0\
	78.9
	21.1

The leach liquors carried the following values in grammes per litre:—

	Copper	Nickel	Iron
1. Water leach.....	9.10	2.28	0.14
2. Acid wash.....	4.85	1.02	1.2
3. Water wash.....	1.72	0.6	0.8

The liquors were evaporated down for convenience in holding over for treatment.

The next trial on matte made from the same ore shows improved results. In this test the chloridizing furnace and the leaching tanks were in operation at the same time. Each batch as soon as withdrawn from the chloridizing furnace was immediately transferred to the leaching tank, thus utilizing the heat of the chloridized matte. Small-scale experiments having shown that the copper extraction was improved by using the saturated absorption water from the chloridizing unit, the leach was purposely confined to this, in order to try out this solvent.

The results of this test are as follows:—

Products	Weight in kilograms	Analysis in per cent			
		Copper	Nickel	Iron	Sulphur
Ore.....	130.0	0.77	2.16	36.42	20.18
Matte.....	81.7	1.02	2.74	43.83	26.21
Roasted matte.....	87.2	1.07	2.91	46.46	8.04
Leached residue.....	62.0	0.13	2.68	62.73	0.91
Percentage of ore in solution.....		92.0	41.0		

The above percentages expressed in the form of ratios show clearly the absence of material loss in the several operations. The ratios are as follows:—

	Nickel — Copper	Iron — Copper- nickel	Iron — Copper	Iron — Nickel
Ore.....	2.8	12.4	47.3	16.8
Matte.....	2.7	11.6	43.0	16.0
Roasted matte.....	2.7	11.6	43.4	15.9
Leached residue.....	20.6	22.3	485.0	23.5

It will be noted that the ratios are fairly constant until the leach is performed.

The ratio of the nickel to iron in the leached residue shows that steel carrying 4.27 per cent nickel may be produced or if the copper is taken into consideration the combined percentage is 4.47.

The residues show considerable improvement over the first test.

The proportion of copper to nickel is 1 : 20.6 which, apart from some trade prejudice is not now considered to be in any way objectionable. The sulphur could be taken care of in steel-making by slagging.

On the other hand the ultimate goal is the total elimination of the copper and sulphur with a higher extraction of the nickel.

Small-scale experiments show that improvements can be made and in the next trial, which is now underway, better results are anticipated.

The residues, used as a pigment, make an iron oxide paint of good covering quality.

Small-scale experiments show that the extraction of nickel can be improved by the use of brine as a solvent.

The leach solutions were combined in a storage tank and are being held for treatment. The wash waters were approximately measured, sampled, and then run to waste. In this trial, no attempt was made to build up the solutions. The analyses are:—

Solutions	Volume litres	Grammes per litre		
		Copper	Nickel	Iron
Combined leach.....	162	4.96	4.3	1.79
First wash.....	68	0.26	0.31	0.27
Second wash.....	71	0.012	0.03	0.03

Small-scale experiments were run on the combined leach solution and these tests indicated that the recovery of the copper and nickel, by electrolysis, will be attended with no great difficulty. It will be necessary, however, to make arrangements and carry on further trials on a scale which will permit possible trouble, not troublesome on a small scale, to develop and be overcome. The following experiments indicate a line that could be followed.

With a carbon anode and copper cathode, 98.2 per cent of the copper in solution was recovered, without attempting an end point, and this was followed by removal of the small amount of iron in solution; the cathode was then changed to nickel and 97.4 per cent of the nickel in solution was deposited. This experiment was repeated in several ways by changing the means for the removal of the small amount of iron, with equally favourable results. A small amount of chlorine is given off but in the above small tests no attempt was made to collect or make use of this. In a large-scale experiment this is one of the features that will be investigated.

Two batches of matte are now ready for treatment, one of 92 kilograms from Murray ore and the other of 25 kilograms from Noranda ore. Small-scale experiments are now being made with these mattes, with a view to taking care of points not attended to in the above tests, and at the same time attention will be given to the precious metal values as well.

It is the intention to combine all the residues and reduce them to steel. The resulting ingot will be subjected to the usual physical and chemical test to determine its qualities.

Conclusions.—The trials reported on indicate the results obtained with the new apparatus. A recovery of 92 per cent of the copper and 41 per cent of the nickel values of the ore has been made in solution, from which they may be easily extracted by electrolysis.

The residues in the form of finely divided oxide represent a recovery of over 96 per cent of the iron value of the ore combined with the remainder of the nickel. The residues are clean and may readily be reduced in an electric furnace to steel carrying about 4.25 per cent nickel. They may also find use in the production of sponge iron or as a pigment.

Experiments now under way have in view the improvement of the above results and the recovery of the precious metal values as well.

V

REPORT ON THE WORK OF THE CHEMICAL LABORATORY
OF THE DIVISION

H. C. Mabee
Chemist-in-charge

During the year 1925, 1,109 samples of ores and mineral products were received and reports issued thereon. This involved about 4,850 chemical determinations. Practically all the samples received were directly connected with the experimental test and research work conducted by the officers of the Division in carrying out their investigations. The usual wide variety of ores and products was received and reported upon, as indicated in the following list:—

Aluminum.....	2	Lithium (mica).....	2
Arsenical.....	16	Molybdenite.....	69
Bronze scrap.....	25	Nickel-copper.....	12
Calcite.....	8	Nickel-silver-cobalt.....	16
Copper.....	8	Platinum-gold.....	7
Copper-silver.....	19	Silver ore.....	17
Copper-zinc-silver.....	53	Silver-gold.....	124
Gold ore.....	117	Silver-lead-zinc.....	16
Gold-copper.....	154	Titaniferous iron.....	8
Lead (metallic).....	2	Samples for identification and	
Lead-zinc-silver.....	412	valuation.....	22

Although these ores represent a fairly wide classification it will be noticed that the greater number are lead, zinc, and copper, with associated minerals, including gold and silver. This is due to the active search for the ores of these metals, samples of which were submitted to the Ore Testing Laboratories for experimental tests.

The above list does not include the determinations made in the electro-chemical laboratory in connexion with the electrolytic iron investigations. In this laboratory a considerable amount of analytical work was performed on iron sulphide and oxide ores, the products from these ores, and on control samples of the test operations.

During the year a technical officer of the National Research Council was given the use of the laboratories for their investigatory work on the beneficiation of Canadian magnesites. Every assistance was given in the way of laboratory space, supplies, etc. The chemical work of their officer is not included in this report.

No change occurred in the permanent staff of the laboratories during the year. B. P. Coyne, R. A. Rogers, and L. Lutes performed the regular routine work, and in addition gave assistance to the investigatory staff in the examination and assay of special samples of products from the test operations.

VI

THE CONCENTRATION OF CANADIAN MOLYBDENITE ORES

INTRODUCTORY

W. B. Timm and C. S. Parsons

Articles have appeared from time to time in the technical press descriptive of the methods applied to the concentration of molybdenite ores in various parts of the world. The chief occurrences in Canada, the history of early concentration methods, the uses, etc., of the metal molybdenum, are fully described by V. L. Eardley-Wilmot in his monograph "Molybdenum" Report No. 592, which has been published by the Mines Branch, Department of Mines, Ottawa.

The present article, prepared in memorandum form, will serve as a supplement to the portion dealing with the concentration of the ores, and covers the practical details of the concentration of the different types of Canadian molybdenite ores. It is intended to assist the mill operator in selecting the type of concentration plant best suited for his ore, and to aid the millman in the manipulation of his concentration so as to attain the best results.

Molybdenite, the sulphide of molybdenum, is the chief mineral of the metal, and is found widely scattered throughout Canada, from Manitoba east to the Atlantic seaboard; and from the Rocky mountains west to the Pacific ocean. During the war quite an industry was developed in Canada; a number of concentration plants were erected, and reduction plants were established at Belleville and Orillia; in 1918 the Canadian production of molybdenite reached a total of 378,029 pounds. The rapid decline in the market price at the close of the war was responsible for the closing down of all Canadian properties. However, the market is now steadily rising due to the depletion of war stocks and the progress made in metallurgical research in finding new uses for the metal, so that the future looks promising for the revival of the molybdenum industry in Canada.

CHARACTER AND TYPES OF MOLYBDENITE ORES

From the viewpoint of the operator and the millman who are concerned with the concentration of the ores, Canadian ores may be classified into the following types:—

1. Ores of the large flake variety in which the molybdenite occurs in flakes above half an inch in diameter, the associated mineral being iron sulphides in a pyroxenite-calcite gangue. As an example of this type may be cited the ores of the Spain mine in Griffith township, Renfrew county, Ontario.

2. Heavy pyritic ores, in which the molybdenite is of the medium flake variety, associated mainly with the iron sulphides, pyrite and pyrrhotite, the gangue being as a rule pyroxenite. In this type of ore the flake is mostly above one-eighth of an inch in diameter, and may be present up to two inches in diameter. As examples of this type may be cited the ores of the Bain mine in Masham township, Hull district, Que., and the ores of the Renfrew Molybdenum Mines, Brougham township, Renfrew county, Ontario.

3. Ores of the medium-flake variety in which the molybdenite is more or less disseminated throughout the pyroxenite gangue rock and associated with small amounts of iron sulphides. The flake as a general rule is smaller than the No. 2 type, requiring finer grinding to free it. As an example of this type may be cited the ores of the Joiner property on lots 3 and 4, concession XX, Cardiff township, Haliburton county, Ontario.

4. Ores of the medium-flake variety in which the molybdenite with an appreciable amount of iron sulphides is disseminated throughout a gangue of quartz, feldspar, and fluorite (altered syenite-gneiss). The flake of this type rarely exceeds half an inch in diameter, and is all freed by grinding to from 35 to 48 mesh. As an example of this type may be cited the Moss Mine ores, Onslow township, Pontiac district, Quebec.

5. Ores of the medium-flake variety; the molybdenite occurring in feldspathic quartz veins with sericite. The molybdenite is generally in the form of rosettes not exceeding half an inch in diameter, and can be freed from the gangue by crushing to 35 to 40 mesh. As an example of this type may be cited the ores on the properties of the Molybdenite Reduction Co., in LaCorne and Malartic townships, Abitibi district, Quebec.

6. Ores of the fine flake and amorphous variety; the molybdenite occurring along the fractures and disseminated throughout the quartz gangue. Examples of this type are the ores of the Alice Arm district, B.C., and the deposit at Kakabeka Falls, Ont. Very fine grinding (to at least 80 mesh) is required to free the molybdenite from the gangue rock.

CONCENTRATION PROCESS BEST SUITED FOR CANADIAN ORES

Molybdenite is one of the easier flotative minerals, as it possesses a marked affinity for oils, especially kerosene, to a similar extent to graphite. Due to its flaky nature and its greasy feel, it is readily concentrated by the flotation process, either by the film or by the froth process. Fair results were obtained by the former by simply floating the dry ground ore on water, or by oiling the wet ground ore and bringing it to the surface of the water. By both these methods the molybdenite floated off on the surface of the water, and the gangue sank. These methods were superseded by froth flotation, due to the greater simplicity of this process, the smaller floor space required for plant, higher grade of concentrates and increased recoveries obtained, and other considerations.

GENERAL FLOW-SHEET FOR CANADIAN MOLYBDENITE ORES

With slight alterations to meet the requirements for each particular ore, the flow-sheet, Figure 5, will be found to give the most satisfactory results with respect to the grade of concentrates produced, recovery, simplicity and low cost of operation.

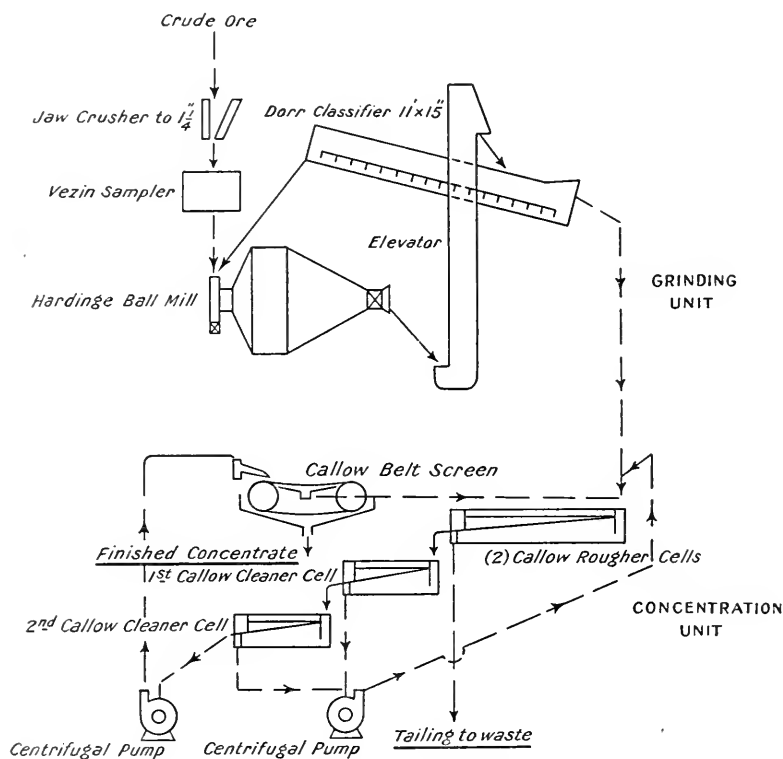


Figure 5. General flow-sheet for the concentration of molybdenite ores.

In the case of ores of the first-mentioned type, the large flakes can be hand-picked, and after crushing the ore is given a preliminary treatment by rolling and screening to obtain as much coarse flake as the operation will warrant. The ore can then be passed to the ball mill and the general flow-sheet followed.

On ores of the second type, a preliminary operation will hardly pay unless a good percentage of the flake is large—above 1 inch in diameter. The general flow-sheet can be followed with few exceptions. More careful manipulation is required to produce a high-grade concentrate on account of the preponderance of iron sulphides present in this type.

For ores of the more or less disseminated type as represented by the third, fourth, fifth, and sixth types described, the flow-sheet can be closely followed with good results. Each particular ore will require certain variations as to grinding, depending at what fineness the flake is entirely freed;

to pulp densities of grinding and flotation operations; to the amounts of reagents used; deflocculation devices, and aperture of the screens used in the final operation.

Most molybdenite ores can be hand-sorted as the mineral is easily distinguished, thus permitting the barren rock to be discarded. Sorting should, therefore, be done whenever possible, so as to give a fair grade of feed to the concentrating plant. As a general rule, the ores can be sorted up to 1 per cent MoS_2 without discarding payable ore, but it would seem that the operation cannot be carried much beyond this point.

CONCENTRATION RESULTS ON THE VARIOUS TYPES OF ORES

Type No. 1

A considerable tonnage of this type of ore was treated during the war when the Ore Testing Laboratories were receiving ores on a custom basis for concentration purposes. As a great number of small individual shipments were received, the concentration results from each were not kept separate. The general procedure with ores of this type was to pick out any large flake that was free of gangue; the ore was then crushed in a jaw crusher, and after removal of the fines any further clean flake was sorted out by passing over a picking belt. The coarse ore was then fed to rolls, and by screening on a 2-mesh screen an oversize product, containing a high percentage of flake, was obtained which was further rolled and screened until the product was of commercial grade. The throughs from the above operations were fed to the ball mill and the regular flow-sheet followed. As a general rule, in ores of this type considerable waste rock can be discarded by hand-picking.

Type No. 2

These ores do not, as a general rule, lend themselves to hand-sorting for the recovery of clean flake. Some flake may be obtained in this manner, but care should be taken not to carry the operation to the unprofitable limit. Barren rock, however, can be discarded and picking operations should be practised to remove waste, so as to give a good feed to flotation. Ores of this type are not so amenable to concentration with the production of high-grade concentrates as the more siliceous types that contain lesser amounts of iron sulphides, due to the tendency of the fine iron sulphides to float and adhere to the molybdenite in the flocculation of the fine flakes. The flake itself in many cases contains fine iron between the laminae, so that concentrates much above 90 per cent MoS_2 cannot be obtained by any mechanical means.

The results of a small-scale test and of a tonnage test on a carload shipment of ore from the Indian Lake deposit, Masham township, Hull district, Quebec, are given below. The ore is of the medium-flake variety in which the molybdenite is associated with a large amount of massive iron pyrite. The gangue is pyroxenite and other lime-silicate minerals.

The richer portions of the ore are almost entirely iron sulphides.

Small-scale Test

Product	Weight		Assay MoS ₂ per cent	Per cent MoS ₂ values	Remarks
	Grms.	Per cent			
Concentrate—					
+80 mesh.....	17.7	1.8	89.25	65.1	} Total recovery, 88 per cent.
—80 mesh.....	37.2	3.7	17.00	26.0	
Middling.....	33.3	3.3	1.94	2.6	
Tailing.....	891.8	89.2	0.17	6.3	4 lb./ton kerosene.
Feed.....	980.0	98.0	2.42	100.0	1 lb./ton pine oil. A little lime added.

Tonnage Scale Test¹

Weight of shipment.....	61,500 pounds
Analysis of sample cut by Vezin sampler.....	0.83 per cent MoS ₂
Average analysis daily sample wet feed to flotation cells.....	0.92 " "
Analysis of feed by calculation from concentration results.....	1.00 " "
Content in ore, assuming 1.00 per cent MoS ₂ as assay of ore.....	616.7 pounds
Weight of concentrates obtained.....	668.5 " "
Average analysis of concentrates.....	78.5 per cent MoS ₂
Content MoS ₂ in concentrates.....	524.60 pounds
Clean-up from run (669 pounds at 2.98 per cent MoS ₂).....	19.9 " "
Average analysis of tailing samples.....	0.12 per cent MoS ₂
Content in tailings, MoS ₂	72.2 pounds
Recovery MoS ₂ from concentration products.....	87.9 per cent
$\text{Recovery figured from formula } R = \frac{100 (H-T) C}{H (C-T)}$	
	88.2 " "

Reagents Used.—Kerosene 0.8 lb. per ton; pine oil 0.3 lb. per ton.

Fineness of Grinding.—12.5 per cent on 65 mesh; 50 per cent on 100 mesh.

Conclusions from Results of Concentration Tests.—The above results show that on an ore of this type averaging 1 per cent MoS₂, a concentrate containing 80 per cent MoS₂ can be produced, with a recovery of better than 88 per cent of the molybdenite values in the ore. The ore submitted was taken from near the surface and was somewhat oxidized. This state of the ore did not have any appreciable effect on the recovery, as tailings as low as 0.08 per cent MoS₂ were produced during the run. It may, however, have had some effect on the grade of the concentrate, and it is possible that on fresh ore of this grade a concentrate of 85 per cent MoS₂ could be obtained. The production of a high-grade concentrate from this type of molybdenite ore in which the iron sulphides predominate over the siliceous gangue minerals, is more difficult than from the more highly siliceous ores. More careful control of reagents, pulp densities, deflocculation devices, etc., is required. With proper control, the results given above should be obtained, with the possibility of producing higher grade concentrates on freshly mined ore.

Type No. 3

The dissimilarity of this type as compared with No. 2 is not in the size of the flake but in the absence of large amounts of iron sulphides. The size of the flake generally prohibits hand-picking but waste rock should

¹ For further details of these tests readers are referred to:—

Mines Branch, Dept. of Mines, Canada, Sum. Rept., 1921, pp. 191-194.

Mines Branch, Dept. of Mines, Canada, "Investigations in Ore Dressing and Metallurgy, 1924," pp. 68-71.

be discarded to give a good flotation feed. The type of ore is very amenable to concentration with the production of high-grade concentrates and good recoveries. The general flow-sheet can be followed very closely. The following results of some small-scale tests, and of a tonnage test, are given on the various classes of ore from the deposits on lots 3 and 4, concession XX, Cardiff township, Haliburton county, Ontario.

Small-scale Test

Four lots of ore were received, representing four types found on the property. In Lot No. 1 the ore was of the medium-flake variety, the flake being much larger and containing more iron sulphides than the other lots. In Lots Nos. 2 and 3, the flake was much smaller being more evenly disseminated throughout the gangue. The ore contained very little iron sulphides. In Lot No. 4, the rock contained an appreciable amount of graphite.

Lot No. 1.....	0.027 per cent MoO ₃	0.68 per cent MoS ₂
Lot No. 2.....	0.027 " "	0.32 " "
Lot No. 3.....	0.025 " "	0.38 " "
Lot No. 4.....	0.025 " "	0.12 " "

Lot No. 4 was too low-grade to be worked commercially and no tests were made.

Lot No.	Mesh	Product	Weight		Assay MoS ₂ per cent	Per cent MoS ₂ values	Remarks
			Grms.	Per cent			
1	48	Concentrate—					
		+80.....	7	0.7	93.31	81.9	Total recovery 91 per cent.
		—80.....	31	3.1	2.20	8.5	
		Middling.....	57	5.7	0.38	2.8	½ lb./ton kerosene.
		Tailing.....	902	90.5	0.06	6.8	½ lb./ton pine oil.
2	65	Concentrate—					
		+80.....	3	0.3	80.25	75.6	Total recovery 84 per cent.
		—80.....	33	3.2	0.58	5.9	
		Middling.....	86	8.5	0.18	4.7	½ lb./ton kerosene.
		Tailing.....	891	88.0	0.05	13.8	½ lb./ton pine oil.
3	65	Concentrate—					
		+80.....	4	0.4	87.25	75.4	Total recovery 83 per cent.
		—80.....	39	3.8	0.64	5.4	
		Middling.....	96	9.5	0.20	4.1	½ lb./ton kerosene.
		Tailing.....	872	86.3	0.08	15.1	½ lb./ton pine oil.

Tonnage Scale Test

This test was on a carload shipment of low-grade ore from the deposit first opened up on the property. The molybdenite flake was up to half an inch diameter in a pyroxenite gangue.

Weight of shipment.....	59,026 pounds
Analysis of ore.....	0.308 per cent MoS ₂
Analysis of concentrates obtained.....	89.65 " "
Analysis of tailings.....	0.115 " "
Recovery of molybdenite values.....	62.7 "

Reagents Used.—0.75 pound per ton kerosene and pine oil.

Conclusions from Results of above Tests.—The ore is very amenable to flotation, with the production of a high-grade concentrate and good recoveries. A 90 per cent concentrate should readily be made from these ores. The large-scale test was made during the earlier test work on molybdenite ores and the best operating conditions were not practised. Tailings as low as 0.05 per cent MoS_2 should have been produced on this grade of ore, which would give a recovery of 83.8 per cent. The small-scale tests which were run later indicate this possible recovery.

Type No. 4

The flake of this variety of ore is usually below $\frac{1}{4}$ inch in diameter and fairly well disseminated throughout the siliceous gangue rock which consists of quartz, feldspar, and fluorite (altered syenite-gneiss). An appreciable amount of iron sulphides is present. The nature of the gangue makes the rock more difficult to crush than the other types but as all the flake is freed at 48 mesh, crushing and grinding costs are reasonably low. The ore is very amenable to concentration so that the general flow-sheet can be followed very closely. Sorting out of waste rock should be practised to give a good grade of mill feed. Ores of the former types appear to contain much more molybdenite than ores of the same grade of this type, due to the smaller flake and the more disseminated character of the latter. The following tonnage tests were made on two carloads of ore from the Moss mine, Onslow township, Pontiac district, Quebec.

Tonnage Scale Tests¹

Shipment No. 1—

One car of ore, net weight.....	50,600.0	pounds
Analysis of sample cut from dry ore by Vezin sampler....	2.21	per cent MoS_2
Average analysis daily sample wet feed to flotation cells.	2.45	" "
Average analysis of dry and wet samples.....	2.33	" "
Content MoS_2 , using average analysis.....	1,178.98	pounds
Amount of concentrate obtained.....	1,213.0	"
Analysis of concentrate.....	92.93	per cent MoS_2
Content of MoS_2 in concentrate.....	1,127.24	pounds
Calculated analysis of tailing.....	0.10	per cent MoS_2
Recovery MoS_2 , from average assay and calculated tailing.....	95.6	"

Shipment No. 2—

One car of ore, net weight.....	84,600.0	pounds
Analysis of sample cut from dry ore by Vezin sampler....	2.33	per cent MoS_2
Average analysis daily sample wet feed to flotation cells.	2.60	" "
Average analysis of dry and wet samples.....	2.48	" "
Content MoS_2 , using average analysis.....	2,081.16	pounds
Amount of concentrate obtained.....	2,135.0	"
Analysis of concentrate.....	93.44	per cent MoS_2
Content of MoS_2 in concentrate.....	1,994.94	pounds
Analysis of tailing, average of daily samples.....	0.12	per cent MoS_2
Calculated analysis of tailing.....	0.12	" "
Recovery of MoS_2	95.8	"

Conclusions from Results of Concentration Tests.—The ore is very amenable to concentration by flotation. With proper manipulation and under the right conditions, a high-grade concentrate, 93 per cent MoS_2 is obtained with recoveries in excess of 95 per cent of the molybdenite content

¹ For further details of these tests readers are referred to:—

Mines Branch, Dept. of Mines, Canada, "Investigations in Ore Dressing and Metallurgy, 1924," pp. 68-71.

in the ore. Concentrates assaying over 96 per cent MoS_2 were made during the runs under the best conditions. The concentrates are exceptionally free from deleterious substances; are suitable for the manufacture of molybdic acid, molybdic salts, molybdenum metal, ferro-molybdenum, and should command the highest market price.

Type No. 5

In this type of ore the molybdenite occurs in quartz, feldspathic quartz, and feldspar veins, generally in the form of rosettes, rarely exceeding $\frac{1}{4}$ inch in diameter. It is usually associated with sericite. Very little iron sulphides are present. The chief known occurrences of this type are south of Amos, Que., and in Nova Scotia. Barren quartz and feldspar are easily sorted out, and this should be practised to give a good grade of ore to the mill. The ores are very amenable to concentration, so that the general flow-sheet can be followed closely. Crushing operations do not need to be carried finer than 48 mesh to free the flake. A tonnage test on the ore from the Molybdenite Reduction Company's properties in LaCorne and Malartic townships, Abitibi district, Que., gave the following results:—

Tonnage Scale Test¹

Weight of ore concentrated.....	19,757.0 pounds
Analysis of ore.....	2.02 per cent MoS_2
Content of MoS_2 in ore.....	399.5 pounds.
Concentrates obtained.....	365.5 "
Analysis of concentrates.....	90.80 per cent MoS_2
Content of MoS_2 in concentrate.....	323.9 pounds
Clean-up of ball mill, etc.....	1,253.5 "
Analysis of clean-up, etc.....	4.59 per cent MoS_2
Content of MoS_2 in clean-up.....	57.54 pounds
Tailings, weight.....	18,147.0 "
Analysis of tailings.....	0.10 per cent MoS_2
Content of MoS_2 in tailings.....	18.15 pounds

Conclusions from Results of Concentration Test.—With an average mill feed of 2.02 per cent MoS_2 , a concentrate averaging 90.8 per cent MoS_2 was obtained with an average tailing of 0.10 per cent MoS_2 . This gives a recovery of about 95 per cent of the molybdenite content in the ore. Higher grade concentrates and lower tailings were made at intervals during the run. The ore is very amenable to concentration. Its physical character and the absence of appreciable quantities of other sulphides such as copper and iron, make it an attractive milling ore. It is not difficult to grind, the molybdenite being freed at about 40 mesh. These characteristics permit of an exceptionally high-grade concentrate being produced with very high recoveries of the molybdenite values in the ore.

Type No. 6

Ores representative of this type are more difficult to concentrate with the production of high-grade concentrates and low tailings. Much finer grinding is required. The fine flake and amorphous variety is generally associated with a quartz gangue. The quartz is sometimes discoloured by the presence of extremely fine flake. Grinding should not be carried beyond the profitable limit. The general flow-sheet can be followed with variations as to degree of grinding, pulp densities, amounts of reagents

¹ For further details of these tests readers are referred to:—

Mines Branch, Dept. of Mines, Canada, "Investigations in Ore Dressing and Metallurgy, 1923." pp. 76-77.

used, and to the screens used for obtaining the finished concentrate. The following concentration results are given from small-scale tests made on ores from Kakabeka Falls, Ont., and Alice Arm, B.C.

Ore from Kakabeka Falls

The molybdenite is of the amorphous variety, filling the fractures in the quartz and disseminated throughout the quartz. A small amount of pyrite is present. Analysis showed it to contain: molybdenite, 2.46 per cent; copper none; bismuth, none; arsenic none.

Test No.	Mesh	Product	Weight		Assay MoS ₂ per cent	Per cent of MoS ₂ values	Reagents
			Grms.	Per cent			
1	50	Concentrate.....	33	3.3	60.97	67.6	1½ lb./ton kerosene.
		Middling.....	38	3.8	9.17	11.6	½ lb./ton crude turpentine.
		Tailing.....	929	92.9	0.68	20.8	
2	80	Concentrate.....	41	4.1	52.44	73.1	3 lb./ton kerosene.
		Middling.....	40	4.0	10.19	13.8	1 lb./ton crude turpentine.
		Tailing.....	919	91.9	0.42	13.1	
3	80	Concentrate.....	40	4.0	58.87	76.6	3 lb./ton kerosene.
		Middling.....	97	9.7	4.93	15.5	1 lb./ton pine oil.
		Tailing.....	863	86.3	0.28	7.9	

Conclusions from Results of Concentration Tests.—The ore requires grinding to at least 80 mesh to give low tailings. The concentrate made from these small tests is below marketable grade but by screening on 100 mesh, a portion of the fine iron and silica adhering to the flakes will be removed. A 70 to 75 per cent MoS₂ concentrate should be obtained from this class of ore. The recovery should be about 80 per cent.

Ore from Alice Arm, B.C.

The molybdenite is of the fine flake and amorphous variety, filling the fractures and disseminated throughout the quartz gangue.

Test No.	Mesh	Product	Weight		Assay MoS ₂ per cent	Per cent of MoS ₂ values	Reagents
			Grms.	Per cent			
1	65	Concentrate.....	31	3.1	66.51	77.6	1 lb./ton kerosene.
		Middling.....	20	2.0	10.72	8.3	½ lb./ton pine oil.
		Tailing.....	949	94.9	0.39	14.1	Little lime added.
		Feed.....	1,000	2.63	
2	65	Concentrate.....	51	5.1	60.45	93.2	1 lb./ton kerosene.
		Middling.....	41	4.1	2.36	2.9	½ lb./ton pine oil.
		Tailing.....	908	90.8	0.14	3.9	Little lime added.
		Feed.....	1,000	3.28	

Conclusions from Results of Concentration Tests.—The ore requires grinding to 65 mesh to obtain low tailings. The concentrates made are below marketable grade but by deflocculation and screening to remove adhering silica and pyrite a 70 to 75 per cent concentrate should be obtained from this class of ore. The recoveries should be between 85 and 90 per cent. The copper content of the ore was about 0.2 per cent, and the concentrates assayed 0.15 per cent copper.

NOTES ON FLOTATION OF MOLYBDENITE ORES

1. The ore should be ground to pass practically all through 35 mesh for flotation. Large thick flakes will be lost in the tailing unless ground to this mesh. The fineness of grinding will depend on the mesh at which the flake is freed and on the character of the flake. A large thin flake will float with more ease than a small thick flake. The degree of grinding should be determined by small-scale laboratory tests.

2. The reagents best suited for the flotation of molybdenite ores are pine oil as the frothing oil and kerosene as the collecting oil. The best quality steam-distilled pine oil, such as the General Naval Stores No. 5, and the British American Oil Company's "Lampolene" brand of kerosene, are preferable. The amounts of these oils used need to be determined for each individual ore, and will depend on the character of the ore, its molybdenite content, the nature of the flake, whether thin, flat, or heavy, thick flakes, etc. For an ordinary clean ore of 1 per cent grade the average amounts of reagents will be: pine oil about 0.25 pound per ton and kerosene 0.5 to 0.75 pound per ton. If an excess of pine oil is added, the flotation circuit will build up with too much dead oil, giving a voluminous froth with very little carrying property, resulting in the lowering of the grade of the concentrate and increasing the loss in the tailing. The oil should be added to the ball mill, especially the kerosene, a portion of the pine oil can be added to the ball mill and a portion to the head of the flotation cells. The addition of other reagents, such as lime and soda ash, is not necessary on an ordinary clean ore, although they have the effect of raising the grade of the concentrate, this is at the expense of a higher tailing. The use of these reagents increases the amount of kerosene necessary to produce low tailings. In cases where the ore is very badly oxidized and the circuit becomes built up with soluble salts, destroying the effectiveness of the flotation oils, it becomes necessary to use alkaline salts to counteract the acidity of the ore.

3. The proper density of the pulp in the ball mill and in the flotation circuit should be determined for each individual ore. The best results are usually obtained with a pulp density of between 45 and 50 per cent solids in the ball mill. The correct oiling effect on the molybdenite to give the best results is not obtained by too thick or too thin a pulp. The proper pulp density can be gauged by the character of the froth on the flotation cells. Using Callow pneumatic cells, the froth should consist of lively breaking bubbles, the molybdenite appearing on the convex faces near their domes, and all coming off in the first foot or so of the cells. If on the other hand the froth is voluminous, heavy, and slow-breaking, and the molybdenite appears in the troughs between the bubbles and is carried down towards the discharge end, the proper conditions do not prevail.

4. In order to produce a high-grade concentrate care should be taken to keep the flake at the coarsest size possible at which it will readily float, and at the same time, free from any attached particles of gangue or other mineral. In order to prevent freed flake from returning with the classifier oversize, the classifier can be equipped with air and water sprays. By permitting the feed to plunge down into the settling area, thus creating a boiling action which tends to bring the flake to the surface; by using water sprays on the rakes to wash out the flake from the oversize; and by the use of an air spray to blow the flake towards the overflow, it is possible

to reduce the amount of flake returning to the mill with the oversize by approximately 50 per cent.

5. In the flotation of molybdenite there should be ample rougher cell capacity to make a clean tailing to go directly to waste, and to provide for any emergencies such as increased grade of the ore, the adjustment of flotation conditions, and the return of large amounts of middling products from the cleaner cells and the finishing screens. The cleaner cell operations should be crowded, so as to take off as high grade products as possible. To obtain high grade concentrate it is better to have two cleaners, one following the other, than only one. A much higher grade product will be sent to the finishing screen.

6. In the case of some ores which are exceptionally clean, the finishing screen can be dispensed with, but as most ores contain some pyrite and other minerals, which in a fine state of division will float and become entangled with the molybdenite, the screen is necessary. The mesh of this screen will depend on the fineness of the flake. For ores other than No. 6 type described above, an 80-mesh "Ton-cap" screen is used. For ores of the No. 6 type, 100-mesh screens are necessary. The use of the finishing screen for increasing the grade of the concentrate depends on the coagulating effect of kerosene on fine particles of molybdenite, thus permitting them to form a larger unit and stay on the screen while the fine pyrite and gangue passes through. A series of analyses made of the concentrate before and after passing over the screen showed that the use of the screen raised the grade 5 to 10 per cent in MoS_2 content.

This coagulating effect, or flocculation, of the molybdenite has a tendency to include some fine pyrite and other minerals with it, and has to be broken up to a certain extent before the cleaner concentrate passes onto the screen. This can be done by feeding fresh water with the cleaner concentrate to a centrifugal pump and pumping to a feed-well for distribution onto the screen. The control of the amount of water is important. An excess breaks up the flocculation of the molybdenite permitting too much fine molybdenite to pass through the screen and be returned to the flotation circuit. Too little will not permit the fine pyrite and gangue to become disentangled, thus lowering the grade of the concentrate. The use of the centrifugal pump between the last cleaner cell and the screen is to re-flocculate the molybdenite in a pulp of clean water, so that the fine pyrite and gangue will be delivered onto the screen free from entanglement in the flocculated molybdenite, and so pass through the screen.

Market Requirements

The market requirements call for 85 per cent molybdenite in ores and concentrates. During the war a concentrate assaying over 65 per cent MoS_2 was received by the reduction plants, provided that the other minerals contained in this low-grade concentrate were mostly iron sulphides. As a rule this was the case, and permitted straight reduction to ferro-molybdenum without the addition of iron, with the production of the ferro-alloy containing 70 per cent molybdenum, and less than 0.4 per cent sulphur. It was on this basis of analysis that the ferro-molybdenum produced in Canada was sold. Molybdenum concentrates should, therefore, contain over 65 per cent MoS_2 for the reduction to ferro-molybdenum to be used for steel purposes. Small amounts of copper and of other metals, below

0.5 per cent, should not be harmful, as the amounts of the metal used for steel purposes is generally under 0.5 per cent molybdenum, so that the amount of copper and other metals taken up by the steel from the ferromolybdenum will be infinitesimally small.

For the manufacture of molybdic acid, molybdenum salts, and molybdenum metal, the concentrates should be as high grade as possible.

Concentrates assaying over 90 per cent and as high as 95 per cent MoS_2 are preferred for these purposes.

Canadian molybdenite ores as a rule are exceptionally free from harmful impurities and the concentrates produced from them are high grade. This is especially true of the eastern deposits. The western deposits are more apt to contain copper and other impurities.

Market Quotations

The New York quotations for molybdenite ores and concentrates are 75 to 80 cents per pound of MoS_2 content in ores and concentrates containing 85 per cent molybdenite and over. Against this price there is a United States tariff of 35 cents per pound molybdenum content, or 21 cents per pound of MoS_2 content, which makes the Canadian market price 54 to 59 cents per pound, less transportation charges. European quotations are 45 to 50 shillings per unit which, reduced to Canadian currency, is 50 to 55 cents per pound, or about equal to the United States price, after the custom tariff is deducted.

Present Outlook

It would seem that for the immediate future, holders of molybdenite properties could not figure on much advance over the present price, namely 50 to 55 cents per pound at Canadian points of shipment. The price of molybdenite is controlled by the price of other metals used for a similar purpose in the manufacture of ferro-alloys. Although there are certain cases where the use of molybdenum has proven its superiority over others of the rarer metals, up to the present this use has not consumed any large amounts. The price will, therefore, be governed by such metals as vanadium and other alloys of steel having a fairly large production. A factor governing the price will be the ability of the United States operators to produce at the present market price. The greatest obstacle to Canadian production at the present time is the United States tariff. Production is being retarded by the uncertainty of the market, and by the different viewpoints of the mine owners and buyers of molybdenite concentrates. The mine owners desire to be assured of a contract for their output at a reasonable price before proceeding with the expenditure of any great amount of money in developing their properties. The users of molybdenum desire to be assured of a uniform and continuous supply at the lowest market price, before changing over to molybdenum for alloy steel manufacture. The revival of the industry in Canada will depend on the assurance of a steady demand for the concentrates at a price indicating profitable operations.

Limitations to Workable Deposits

A 1 per cent molybdenite ore is a \$10 ore with molybdenite at 50 cents per pound. On a basis of 50 tons a day, or even as low as 25 tons a day, operating expenses should not exceed \$7.50 per ton, on any fair-sized deposit. In the case of one company operating during the war, it was shown that with molybdenite at \$1 per pound, the workable limit was 0.3 per cent MoS_2 , on a basis of 100 tons daily. As shown from the concentration of the ores given above, the process is simple, the majority of Canadian ores are readily amenable to concentration, grinding in most cases does not have to be carried finer than 48 mesh, so that concentration charges are low. The chief item of cost is mining and development. Owing to the character of the deposits and veins, considerable waste has to be broken and sorted out. Much development has to be done in waste rock, so that mining costs will be as high, if not greater than lode mining. Capital expenditure for mining plant and equipment will be similar to lode mining for the same tonnage mined, but the cost of the concentration plant will be lower, as the process as a general rule is simpler.

THE CONCENTRATION OF CANADIAN FLAKE GRAPHITE ORES

C. S. Parsons

INTRODUCTORY

The recent revival of interest in graphite due to improved market conditions has prompted the writing of this article. The history of graphite mining in Canada should be carefully studied by any company contemplating entering this field. Many failures in the past have been due to inefficient methods of concentration, but a number have resulted also from the erection of extensive milling and plant equipment before the extent and grade of the ore-bodies were determined; and from insufficient knowledge of the market and trade requirements.

TYPES OF CANADIAN GRAPHITE ORES

Two main classes of graphite ores are known to occur in Canada: disseminated crystalline flake graphite, and the more or less massive crystalline graphite which occurs in veins or pockets. The deposits of the latter variety which is commonly called "crystalline graphite," have never been considered commercially important, and for this reason their concentration will not be discussed in this article.

In Canada the average run-of-mine ore of the flake variety will contain 10 to 15 per cent graphite; the average American ore of this type contains only between 3 and 7 per cent.

FACTORS GOVERNING THE VALUE OF A GRAPHITE ORE

In determining the value of a graphite ore, granted that there is a sufficient tonnage available, the governing factor is the size of the flake. Since there is a ready market only for large flake, the value of the ore depends on the amount of No. 1 and No. 2 flake recoverable rather than on the total graphite content of the ore. It is possible that a 10 per cent ore may be more valuable than a 15 per cent ore if more No. 1 and No. 2 flake can be recovered from it. However, certain ores may contain more large flake than others but still be of less value, due to the hardness and nature of the associated gangue. A hard siliceous gangue is difficult to free, causing a large amount of flake to be ground up in the process of separation.

CONCENTRATION

Before the introduction of froth flotation, the concentration of graphite presented many difficulties. It may be conservatively stated that the losses in the tailings exceeded 50 per cent. The actual production of a high-grade graphite concentrate by flotation would be a simple matter, as it is one of the most easily floated minerals, were it not for the fact that the trade demands certain qualities, and pays the highest price for No. 1 flake (which is flake remaining on a 90-mesh screen). The finer sizes have a limited

market and at most times it is impossible to dispose of them at any price. The main problem is, therefore, the design of a flotation plant that will destroy as little as possible of the large flake in the ore. The efficiency of the mill will depend more on the amount of large flake produced than on the total recovery of graphite.

REFINING

It can be readily understood that there is a point beyond which it would not be practical to raise the grade of a concentrate produced by continuous operation in a concentrating mill. Therefore, to determine when concentration should be stopped, and refining in batch lots resorted to, is essential. In the refinery each operation is under exact control and the size of the flake is more easily preserved.

GENERAL PRINCIPLES IN DESIGN OF CONCENTRATING MILL

There are a few general principles which should be kept in mind when designing a flow-sheet for a mill to concentrate graphite ore. The ore should be crushed only to the point where it is possible to obtain a reasonably clean tailing. The concentrate produced by the primary flotation should contain a minimum of free gangue. In recleaning this concentrate the middling products will consist of true middlings and particles of gangue which have become mechanically entangled with the concentrate. The greater part of the entangled gangue can be eliminated by operating the first cleaner cells in closed circuit with the roughers. The flow-sheet should be flexible enough to grade the true middlings into two general classes. These two classes are: first, particles of gangue to which are attached small pieces of graphite; second, particles of graphite to which are attached smaller pieces of gangue. These two classes naturally grade into each other. Just where the division between these two classes should be made will depend on final results in the refining department. After the division has been effected, the first class should be returned to a separate grinding circuit, and the second class treated as final concentrate and delivered to the refinery, where the work of freeing the attached pieces of graphite can be effected more efficiently.

FLOW-SHEET OF AN ALL-FLOTATION PLANT

A flow-sheet of a 200-ton mill which operated three years on a hard, 7 per cent ore is shown as Figure 6. This mill was one of the first flotation plants to treat a hard, siliceous graphite ore. Flotation replaced a system of buddle concentration. A study of this flow-sheet will show that the entire concentrate from the flotation cells is passed through two pebble mills for regrinding. When the mill was first built only one regrinding mill was used, but the additional recovery of graphite overloaded the capacity of the refining plant, and an agitation was immediately started by the refinery superintendent to have a higher grade product delivered to the refinery. A second regrinding pebble mill was then installed. The graphite from this mine was used entirely for lubricating purposes, and even a grade of 88 per cent carbon had to be further refined.

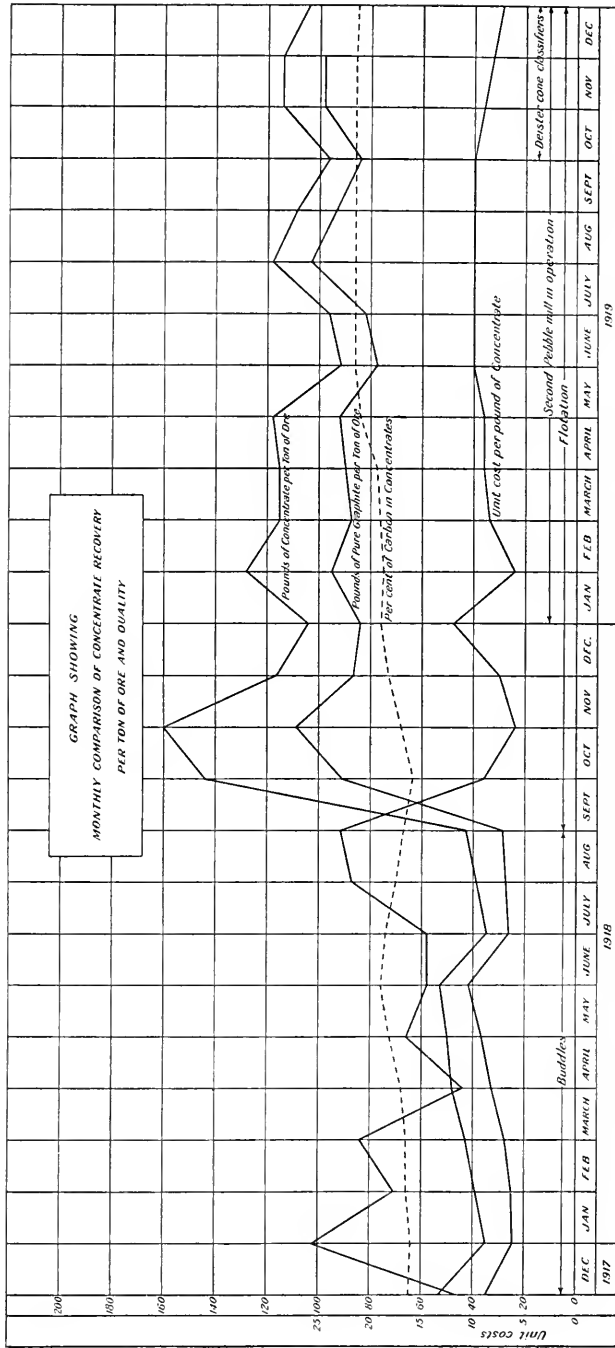


Figure 7. Graph showing comparison of costs using different systems of concentration.

A series of curves is given in Figure 7, showing a comparison between the buddle system, flotation using one regrinding mill, and flotation using two regrinding mills. These curves give the number of pounds of pure graphite (carbon) recovered from a ton of ore; the weight of concentrate recovered from a ton of ore; the grade of concentrate and percentage of carbon contained; and the unit cost per pound of concentrate produced. A comparison between the use of buddles and flotation with reference to the amount of +60-mesh pure graphite (carbon) is given in Table I.

TABLE I

A Comparison of Average Results Obtained on a 6 to 7 per cent Hard Graphite Ore

	Buddles	Flotation —one re- grinding mill	Flotation —two re- grinding mills
Pounds of concentrate recovered from one ton of ore.....	45	120	116.0
Per cent recovery on 60-mesh screen.....	60	43	25.0
Per cent carbon content of +60-mesh product.....	60	73	88.0
Pounds of pure graphite recovered, + 60 mesh.....	16	35	25.5

The flow-sheet used in this mill (Figure 6) is not in accordance with the principles previously given. The clean, coarse flake should have been separated from the concentrate before passing to pebble mills for regrinding. One of the most practical methods of doing this is to table the concentrate first. This flow-sheet is given to illustrate how readily any additional grinding will destroy the coarse flake, and to draw attention to the remarkable improvement in recovery and lowering of costs by flotation.

FLOW-SHEET OF PLANT USING FLOTATION AND TABLING

A flow-sheet, Figure 8, is given of a mill in which tables were used on the flotation concentrate. In this case the sands and part of the middlings were run to waste in the table tailing, instead of being reground and returned to the circuit. The grade of concentrate delivered to the refinery depended largely on the operation of the tables, so that in order to raise the grade it was necessary to turn more graphite into the table tailing, thus causing a loss of considerable large flake. The recovery was low, probably not exceeding 75 per cent, but this was offset by the large proportion of No. 1 crucible flake recovered. It will be observed from the flow-sheet that the flotation concentrate was not reground as in the previous one, so that although the recovery was low, the proportion of No. 1 crucible flake produced was large and the refinery actually produced 70 per cent No. 1 flake of a grade exceeding 92 per cent carbon.

THEORETICAL FLOW-SHEET

A flow-sheet, Figure 9, has been worked out according to the general principles of design given. This flow-sheet starts with grinding in closed circuit with either an Akins or Dorr type classifier, the grinding being carried only to the point where clean tailings can be made from the rougher

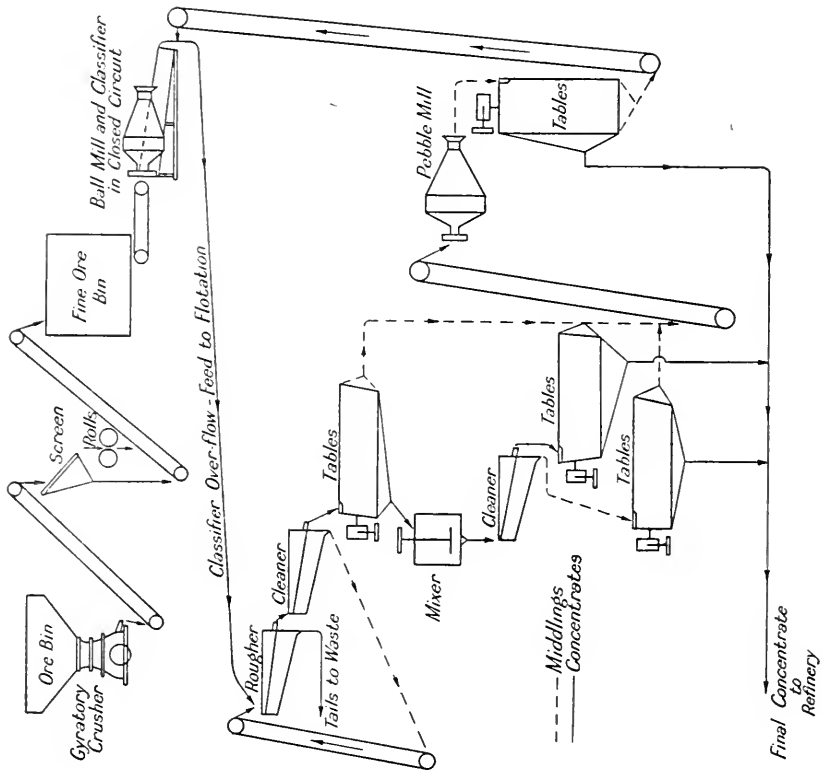


Figure 9. Theoretical flow-sheet for flake graphite ores based on experiments.

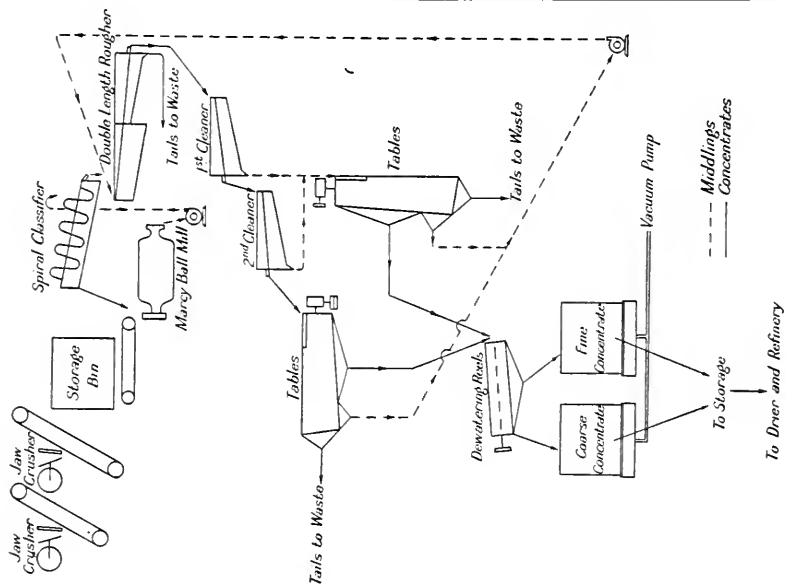


Figure 8. Flow-sheet for graphite mill using flotation and tabling.

cells. There is a possibility that it would be practical at this stage to table the classifier returns to prevent the coarse flake contained in the oversize again passing through the ball or rod mill. The rougher concentrate is cleaned once in a cleaner cell and the cleaner tailing returned to the rougher feed. The cleaner cell should be operated to drop only entangled sands. The concentrate from the cleaner cell goes to tables which as far as possible should be operated to separate only the sand particles containing relatively small flakes of graphite. The concentrate from the tables is dewatered and sent to a second cleaner cell, while the table tailing goes to a pebble mill for regrinding. The second cleaner cell will drop some coarse, free flake as well as flake of various sizes with attached pieces of gangue. The second cleaner tailing is then tabled and a finished concentrate produced. By this tabling operation the clean coarse and fine flake dropped from the flotation cell is recovered and not returned for further grinding in the pebble mill, whereas the flake of various sizes containing attached gangue is sent to the pebble mill for regrinding. The second cleaner flotation concentrate is again tabled; this tabling may not be actually necessary, but it acts as a safeguard against the inefficient operation of the flotation cells. The discharge of the regrinding pebble mill is tabled, and a finished concentrate made, and the sands and entangled particles of graphite and gangue are returned to the head of the rougher cells.

It is believed that a flow-sheet designed along these general lines will give the most efficient results on flake graphite ores. On soft and easily disintegrated ores it will not be necessary to carry the flotation and tabling so far. The particular advantage claimed for this flow-sheet is that besides eliminating the coarse flake from the circuit as soon as it is freed, the fine flake produced in the refinery will be of a much higher grade than that obtained from the two flow-sheets previously given.

INTERPRETATION OF SMALL-SCALE TESTS

The correct interpretation of the results of small-scale laboratory test work requires explanation. The scope of this paper does not allow a lengthy discussion on this subject, but it has been thought advisable to draw attention to the great difference in results obtained by crushing in small laboratory ball mills and crushing in large standard sized mills. One example will be given to illustrate this difference. An exhaustive number of laboratory tests were run on a certain ore to ascertain the quantity of large flake that could be recovered. These tests showed a recovery of more than 50 per cent of the flake on a 60 mesh screen, the product assaying 85 per cent carbon. The results from the mill designed from these tests showed a recovery of only 30 per cent on 60-mesh, assaying 85 per cent carbon. This large difference was due to the action of grinding in large ball mills.

SOME NOTES ON THE TYPES AND OPERATION OF MACHINERY IN GENERAL USE

Grinding.—The ball mill is in general use, and there is little doubt that this is the most practical and satisfactory machine for the final reduction of the ore. The conical type of mill is most favoured because it is claimed that the flake when freed floats out of the mill more quickly than with the cylindrical types. However, the writer has seen equally as

good results obtained in the ordinary cylindrical mill when used with either a diaphragm or open trunnion discharge. The lubricating nature of graphite has been found to reduce the rated capacity of a ball mill 25 per cent, a factor that should be taken into consideration when determining the size of the mill to install. From extensive tests carried out by the writer it was found that less flake was destroyed by grinding in a dilute pulp of from 45 to 55 per cent solids (as much as 10 per cent additional flake reporting on a 60-mesh screen). The rod mill is suggested as possibly giving more efficient grinding than the ball mill. The Ore Dressing and Metallurgical Division of the Mines Branch contemplates carrying out a number of comparative tests with the rod mill and ball mill.

Closed Circuit Grinding with Classifier and Ball Mill.—An excess of water must be avoided in order to obtain efficient results in a flotation plant. Hydraulic classifiers should not be used in the circuit. It has been claimed that they are better than the Dorr or Akins types for preventing the return of the large flake with the oversize to the ball mill, but it has been found in three different mills that any advantage gained in this way is offset by the increased loss of coarse flake in the flotation tailing, due to high dilution of the pulp in the cells.

Flotation Machines.—The Callow pneumatic cell has been more extensively used than any other for the flotation of graphite, but there is no reason why just as satisfactory results cannot be obtained from any other standard type of machine. The type of Callow cell found to be the most satisfactory for graphite is the short 8 by 2 feet cell with sloping bottom. A short, narrow, flat-bottom cell of the same size should also work well. The long, 19 by 3 feet flat-bottom cell or the 40 by 3 feet cell, are not recommended for the average Canadian flake ore.

In the operation of a pneumatic type of cell it has been found advantageous to use a system of baffling in the cell. The sides of the cell are built up with boards to a height of 12 to 14 inches, and a series of three vertical cross baffles of different heights are built in the cells so that the froth descends in cascades and flows through an outlet. In the rougher cell, the outlet is at the feed end, the side boards being cut away 12 to 20 inches from that end, allowing the froth to discharge over the sides. In the cleaner the froth is carried direct through an opening cut in the front or discharge end of the cell. These baffles are carried down to within 6 inches of the blankets and are arranged on top so that there is approximately a 3- to 4-inch drop between each baffle. The use of the baffles makes the operation of the rougher cell much simpler, and by their use in the cleaner cell, the grade of concentrate can be raised 5 to 10 per cent.

Flotation Reagents.—Kerosene and steam distilled pine oil are the two oils which should be used. The use of kerosene, however, requires careful manipulation, as the least excess will tend to kill the froth in the cells. For this reason some operators have had trouble using it, but the writer has operated with it for a number of years and never experienced trouble. Hardwood creosote oil has been used along with the kerosene to overcome the raw oil effect of the kerosene, but its use is not to be recommended, as it is accompanied by a greater loss of graphite in the tailing. Additional reagents such as lime and soda ash are seldom necessary.

Tables.—Any table made by a reputable manufacturer should be satisfactory. Some operators use fine silk bolting cloth over the launder along the low sides of the table, for dewatering the graphite concentrate. The frame holding the silk is attached to the side of the table so that the bumping action of the table keeps it free. In most cases the riffing of the table will have to be altered to take care of the type of feed to the table.

Filters and Dewaterers for the Concentrate.—Continuous suction filters have not as a rule been satisfactory, due to the physical character of the graphite. In most cases it has been found impossible to reduce the moisture content of the concentrates below 30 per cent. The lubricating qualities of graphite, together with the tendency of the thin flat flakes to seal the pores of the filter cloth prevent a cake from being built up and held in the filter until the proper time for discharging. A number of more recent tests made by the United Filters Corporation on a sample of concentrate from one of the Buckingham mills have shown more favourable results. The results of these tests are as follows:—

Test No.	Pulp ratios by weight	Vacuum on cake, inches	Vacuum on dry, inches	Moisture in cake, per cent	Cake thickness, inches	Capacity, sq. ft. per hour, dry graphite
1.....	3 : 1	18	18	20.9	2	216.0
2.....	8 : 1	11	16	20.5	1½	36.0
3.....	20 : 1	15	18	20.5		24.0
4.....	16 : 1	8	17	20.1	9/16	16.0
5.....	5 : 1	5	18	20.6	¾	42.5
6.....	10 : 1	21	20	22.5	¾	60.0
7.....	20 : 1	21	21	24.5	3/8	23.8

In each test, with the exception of No. 4, the pulp in the filter was air agitated. The hourly capacity of the filter varies, due to the speed at which the filter is revolved, the vacuum at which the cake is built, and the pulp ratio. It has been suggested that if the +90 flake were screened out before filtering, much better results could be obtained. The suggestion is practical, and it would appear that better results can be expected.

The system most generally used is a series of settling tanks which are allowed to fill alternately. After settling, the water is decanted and the graphite allowed to drain through a diaphragm of burlap, or other porous material. There is no reason why a Dorr thickener cannot be used. The only objection to the Dorr tank that the writer has heard, is that a heavy froth forms on the top of the tank, but this can be taken care of by building a high retaining ring inside the tank and by occasionally skimming off this froth, which in some cases consists of very high-grade fine flake.

Driers.—The type of drier commonly used is the direct-fired rotary. There are two disadvantages to a drier of this type: contamination by ash from the fuel, and the dust loss. The dust loss can be avoided by the use of dust collectors. In one instance, the installation of a dust collector saved 1,200 pounds of graphite a day. Indirect types of driers, such as the Lowden, or Ruggles-Cole, although the first cost is high, are probably the more satisfactory.

Finishing Plant.—The process of refining must be developed for the particular concentrate to be treated. The most important factors which will affect the treatment scheme to be used are such physical characteristics as, diameter, thickness, hardness, and toughness of the flake, and the nature of the impurities to be removed, whether soft or hard. Coarse, hard impurities are more difficult to handle and the amount of work the refinery should do on this class of product should be balanced against the amount of work which can be done effectively in the concentrating mill. A good deal of information has come to the writer's attention pertaining to different methods of refining, but the information obtained is not considered reliable enough to draw definite comparative conclusions. One large American company which has been in the graphite industry from the beginning, uses a system of batch ball mills and dry screening. A large, successful Canadian company uses burrstone mills and dry screening. Both claim they have tried other methods, and that their present practice has proven the best. The writer favours the use of the burrstone mill. Both the operation of the ball mill and the burrstones must be in the hands of experienced and capable men, as the personal element governs the efficiency of the operation in both cases. An experienced operator will recover 50 per cent more No. 1 flake than an inexperienced one.

The writer recommends that when considering a new installation for a finishing plant, a simple system of burrstones and screens of the Hummer type be tried at first. Any elaborations on this system can be developed gradually as experience with the particular ore shows the way. It will be found advisable to house the finishing plant and drier in a separate building from the concentrating mill, on account of the dust from the dry process of finishing.

THE ORES OF WESTERN QUEBEC—THEIR CHARACTER AND METALLURGICAL TREATMENT

W. B. Timm

The more recent developments in western Quebec have shown the ores of this district to be rather complex in character, requiring more extensive and more difficult metallurgical treatment than was at first expected. The first impression obtained from the initial discoveries, was that the ores of the extension of the Kirkland Lake and Porcupine belts in Quebec would be similar in character to the Kirkland Lake and Porcupine ores. Developments have since shown that although the ores along the southern margin of the belt are only slightly different, the ores of the more recent discoveries within the belt are vastly different and more complex in character, requiring a different metallurgical treatment.

CLASSIFICATION OF THE ORES

From a metallurgical viewpoint the ores may be classified into the following types:

Ores in which iron pyrite is the predominating sulphide. In this class of ore the pyrite is generally disseminated throughout the vein matrix and the gold values are closely associated with it. Small amounts of chalcopyrite may be present but as a rule to no great extent.

Ores in which arsenopyrite is the predominating sulphide. In this class of ore the gold occurs in the fractures of the quartz and is associated to a lesser extent with the arsenopyrite. Iron pyrite may also be present.

Ores in which pyrrhotite is the predominating sulphide. In this type is found a number of different minerals, depending on the degree of replacement of the pyrrhotite by chalcopyrite, pyrite, and zinc blende. On the same property ore-bodies are found ranging from massive pyrrhotite with very small amounts of chalcopyrite, to those in which the chalcopyrite predominates; others of disseminated ore, with chalcopyrite and pyrrhotite in their associated gangue rock; in some others zinc blende predominates. As a rule, the gold values are not associated to any extent with the pyrrhotite but with the other sulphide minerals, chiefly the chalcopyrite. It is this latter type of ore and its sub-types that are of prime importance, the abundance of which is the reason for the establishment of a smelting industry in the district.

EXAMPLES AND METALLURGICAL TREATMENT

The ores of the Arntfield group in Boischatel township and adjacent claims and those of the Huronian Belt Company's claims in Rouyn township, may be taken as examples of the first type, namely those in which iron pyrite is the chief sulphide mineral. This type of ore should present no very difficult metallurgical problem. If the gold values are consistent in being associated with the iron pyrite it would seem that amalgamation followed by close table concentration would be the method applied to

their treatment. The concentrates could be reground and cyanided or shipped to a smelter. Should these ores contain copper values, table concentration could be supplemented by flotation and the entire concentrates shipped to a smelter. Should the gold values be to some extent associated with the gangue minerals the above process would have to be supplemented by cyanidation, providing there is remaining after concentration sufficient values to warrant this process.

The ores of the Edna Bathurst claim in Rouyn township, of the O'Brien mine in Cadillac township, of the E. J. Thompson group in Cadillac township, and of the Sisco mine in Dubuissou township, may be taken as examples of the second type in which arsenopyrite is the predominating sulphide. The gold occurs in blue quartz along the fractures, and to some extent associated with the arsenopyrite and pyrite. The recovery of values from this type of ore should not be difficult. The presence of so much free gold will make amalgamation a primary step in the process of treatment. Whether the ore will be concentrated to obtain an arsenopyrite concentrate before cyanidation or cyanided directly after amalgamation will depend on the market for arsenic, and on the effect of the presence of arsenopyrite on cyanide operations. Some arsenopyrite ores are amenable to cyanidation, others are more refractory.

As examples of the third general type in which pyrrhotite is the predominating sulphide mineral, may be cited those ores which are commanding so much attention at present. These include the ores of the Horne mine, Rouyn township; the Amulet, and the Waite-Montgomery mines, Duprat-Dufresnoy townships; the Alderson-MacKay and Chance groups, Boischatel township; the Abana group, Desmeloizes township; and a number of others. On some one or other of these claims may be found ores with the following characteristics:—

Heavy sulphide ores, ranging from massive pyrrhotite with little chalcopyrite to high-grade copper ore (chalcopyrite predominating over the pyrrhotite). Within this range the high gold values do not always follow the high copper values, but in many cases occur in the ores of the low or medium copper content.

Disseminated ores, where the minerals, chalcopyrite, pyrite, and pyrrhotite, are disseminated throughout their associated gangue rock.

Heavy sulphide ores, ranging from high-grade copper ore (chalcopyrite predominating over the pyrrhotite) with little or no sphalerite to high-grade zinc ores (sphalerite predominating over the pyrrhotite) with small amounts of chalcopyrite. Here again within this range, the higher gold values do not always occur with the rich copper ores.

Disseminated ores of chalcopyrite, sphalerite and pyrrhotite, these minerals being disseminated throughout their associated gangue rock.

From the above classification of the ores of the third general type it will be seen that their treatment will present to the metallurgist many difficult problems. Almost all the ore-bodies or deposits vary in some respect from one another in the character of their ore, so that each will require careful study to determine its amenability to the general scheme of treatment of the ores as a whole. The general scheme of treatment for the ores of this type will no doubt embody the erection of concentrating plants at the various mines with a custom smelter favourably located to receive ores and concentrates.

The concentrating ores will consist of the disseminated ores of the district, and ores of the heavy sulphide sub-types, low in copper, or containing both copper and zinc values requiring the separation of the minerals. These will constitute in time the greater bulk of the ore mined, and to meet this situation the smelter will be required to take a considerable tonnage in the form of flotation concentrates.

The concentrating ores may be summarized as follows:—

Disseminated ores of chalcopyrite and pyrrhotite with, perhaps, a certain amount of pyrite. These will require concentration to separate the chalcopyrite and pyrite, which as a rule carries considerable gold values, from the pyrrhotite and worthless gangue rock. This is a problem of gravity concentration and flotation, or vice versa, or simply straight flotation to recover the copper and gold values in a concentrate suitable for smelting operations. Certain of this class of ores containing a considerable excess of free silica are found in the district, and such ores will be in demand for converter flux, and are therefore more valuable in the raw state than in the concentrated form.

Disseminated ores of chalcopyrite, sphalerite, and pyrrhotite, with perhaps, a certain amount of pyrite. These will require concentration to separate the various constituents into three products, namely, a copper-gold concentrate of the chalcopyrite and pyrite, if it be present, a zinc concentrate, and a pyrrhotite-gangue tailing. This is a type of problem in respect to the solution of which selective flotation has made such rapid strides in the last few years, that there would seem to be no serious difficulty in applying it to this class of ores.

Heavy sulphide ores, chiefly pyrrhotite, low in chalcopyrite, and containing in some cases considerable pyrite and also considerable gold values, associated with the chalcopyrite and pyrite. In the case of ores of this class the economic point between the direct smelting and concentrating of the ores will need to be determined. This will probably lie in the vicinity of 4 per cent copper. Whether concentration is feasible will also depend on the recovery of the gold values. Where the ore is pyrrhotite and chalcopyrite, a separation of these two minerals can be made with the production of a copper concentrate carrying the gold values, with good recoveries in both copper and gold. Where a considerable amount of pyrite is present and this mineral carries a certain percentage of the gold, the separation is more difficult.

Heavy sulphide ores containing chalcopyrite, sphalerite, and pyrrhotite, with perhaps, a certain amount of pyrite. With ores of this class the determining factors between smelting and concentrating ores will be the zinc content in the copper ore, and the copper content in the zinc ore. Between these two points the ore will require concentration, and the method most likely used will be selective flotation making a copper concentrate, a zinc concentrate, and a pyrrhotite tailing. Small-scale tests have shown very favourable results in the concentration of this class of ore.

SMELTING OPERATIONS

The discovery of what would seem to be large tonnages of smelting ore, within a comparatively small radius, has decided those directly interested to establish a smelter in the district favourably situated to the

principal deposits. Concessions with regard to townsite, smelter site, and smoke nuisance seem to have been definitely settled agreeably to the operators and the Provincial Government. The design of the smelter is in progress, if not complete. The railway from O'Brien, on the Canadian National railway, will be ready by the end of the year, when the equipment will be brought in and construction work started. The first unit is reported to be of 500 tons, although later reports, no doubt due to the favourable developments within recent months, have increased this to 1,000 tons daily capacity. The product of the smelter will be blister copper, which it is the intention to ship to the eastern states for refining, at least until refining works are established in eastern Canada near a source of cheap hydro-electric power. For the reduction of copper-gold ores and copper-gold concentrates, two methods of treatment may be successfully applied, namely blast furnace and reverberatory smelting. A careful study of the character of the ores, the percentage of coarse ore to fines and concentrates, the tonnage of each to be treated, available ore reserves, installation, and treatment costs, etc., are points to be considered in determining which of these two processes is the more economical method. It is very probable that both blast furnace and reverberatory smelting will eventually be used, the blast furnace being used for the heavy pyritic coarse ores and the reverberatory for the fines and products of the concentration plants, flue dust, etc. The blast furnace and reverberatory matte will be blown to blister copper in basic converters. Highly siliceous ores containing copper and an excess of free silica, such as the Chadbourne ore, will be used for converter flux.

The commencement of operations may only include a blast furnace plant, advantage being taken of the sulphur content in the heavy sulphide coarse ore to supply a great deal of the heat for reduction. Pyritic or semi-pyritic smelting may first be practised, using a minimum of coke, around five per cent, for reduction. Lime rock for fluxing purposes will have to be brought in from the most favourably situated localities with respect to freight rates, and its content of available free lime. Limestone, coke and coal will furnish incoming freight for the railway.

If the initial plant consists of blast furnace smelting, as great a burden of the heavy coarse sulphide ore as possible will be carried. The fines broken from mining operations, the concentrates from the concentration plants, and the flue dust will require sintering to agglomerate the fine particles together for blast furnace operation. The more of this sinter that is used in the charge, the more coke will be required for reduction. If reverberatory smelting is practised, the fines, concentrates and flue dust with the proper amount of flux necessary to give a free flowing slag, will be pre-roasted in furnaces of the McDougal type, and the calcines fed directly to the reverberatory fired with pulverized coal.

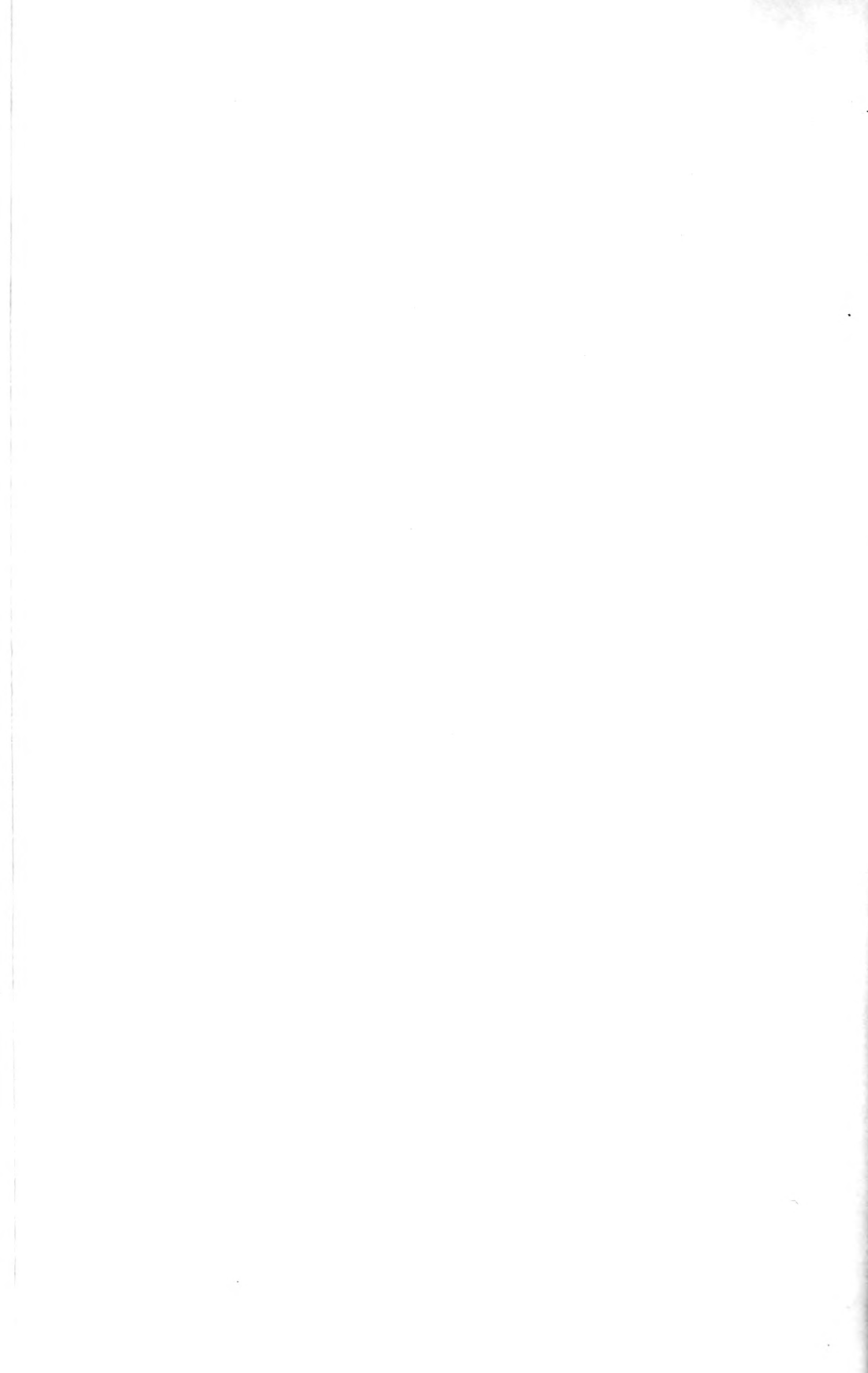
It would seem that some system of banking the various types of ores and concentrates will be necessary to obtain uniform operation. The smelter operator must know the composition of the various ores well ahead, so as to make up his charge and keep it uniform from day to day, otherwise, he will experience all kinds of trouble. This is of especial importance in blast furnace operations where a slight variation in the alumina-iron content of the ores causes trouble in the otherwise easy operation of the furnace. A custom's smelter for treating the ores and concentrates of the district will be called upon to purchase all types of

ore. Some will contain fairly high zinc values and the percentage of this metal in the charge will have to be kept constant; others will contain arsenopyrite to some extent; all these points make the banking system necessary for most efficient operations.

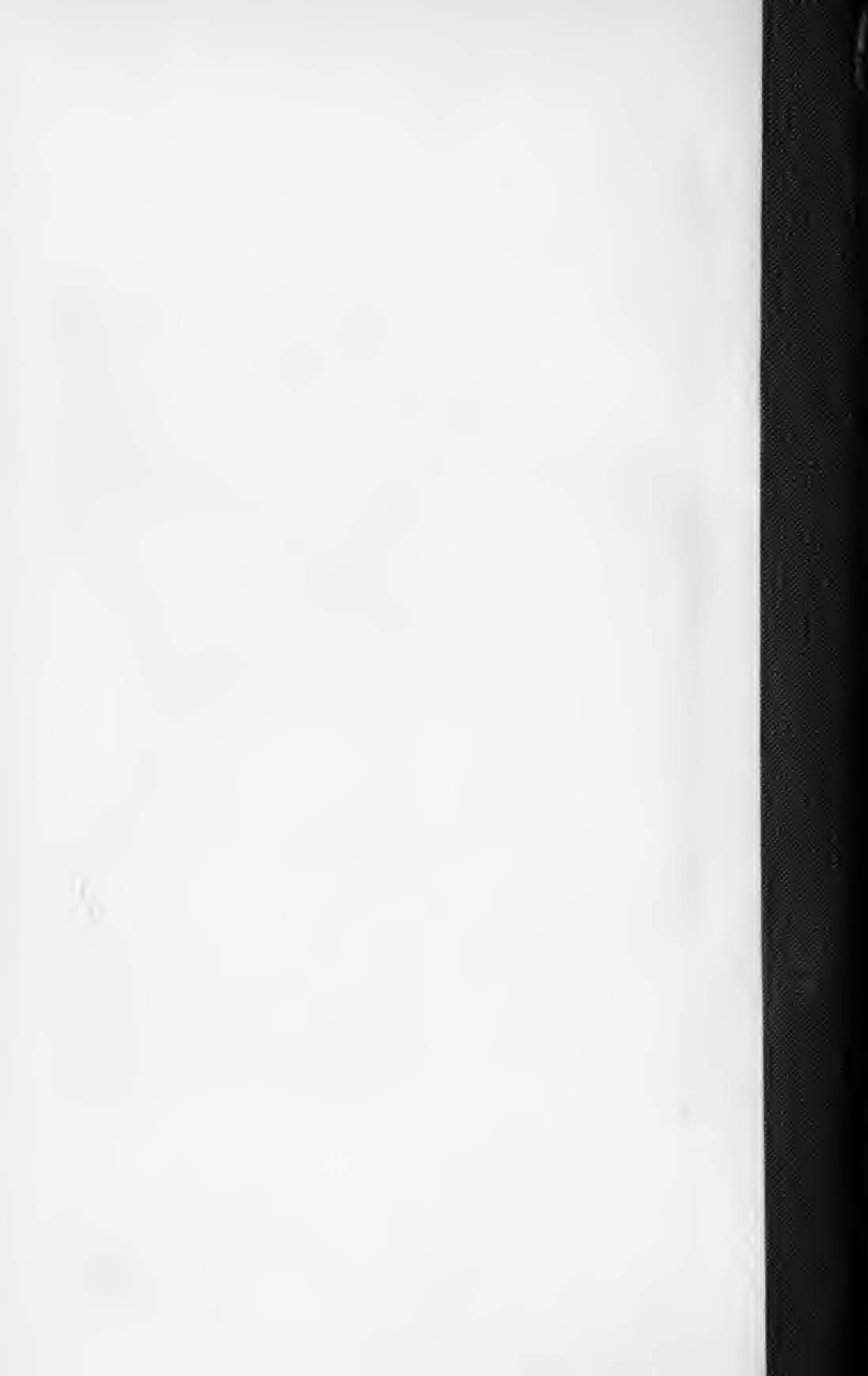
Little so far has been reported on the establishment of reduction works for the treatment of zinc ores and concentrates. This is probably due to the fact that no very large bodies of zinc ore have been opened up. Recent discoveries, however, indicate that sphalerite will be one of the commercial minerals of the district and is present in varying amounts in a number of the deposits. In some cases it would seem to be in commercial amounts, necessitating its separation from the copper and its treatment in a zinc reduction works. It is not likely that zinc ore and concentrates will be treated in the district, probably they will be shipped out to works in the Eastern States or to Belgium, or to a source of cheap power on the St. Lawrence, or Ottawa rivers, or some other favourable point where the electrolytic process for the recovery of the metal in the form of spelter can be more successfully operated. Rumours have been current of the establishment of a zinc refining plant on the Saguenay in connexion with the power development there. If such works materialize it would offer a favourable point for disposal of this class of ore and concentrates from western Quebec. There is also a brisk demand for zinc ore and concentrates to supply the needs of the Belgian works, to which Canada shipped a large quantity of its zinc concentrates during 1924.

SUMMARY

It would appear that the vicinity of the important copper-gold discoveries in western Quebec gives promise to be the centre of a large metallurgical industry where the ores of the district will be reduced in smelting works, the establishment of which will assist materially in the opening up of the country. It will be an incentive to greater mining activity in providing the small operator with a smelter to which he can ship his high-grade ore while his property is in course of development. It will open up a market for farm produce to supply the needs of the mining communities and thus enhance agricultural development in the vicinity. Lumber will be required for construction and mining purposes. In order to save the pulpwood in the vicinity of smelter works, it will require to be cut as soon as possible. Although the character of the ores present difficult metallurgical problems they are not greater than have been surmounted in the past. One has only to review the history of such mining camps as Sudbury, Cobalt, Rossland, Kimberley and others to see evidence of the success made by the operating companies on ores which seemed equally as difficult of treatment, if not more so, than the Quebec ores. In each case the ores have been successfully treated, and moreover, in the case of the Quebec ores, much valuable information on the treatment of similar ores in other parts of the world is already available.







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